Chapter 4

Land Use and Air Quality

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R645-301-410 Land Use

Co-Op mining property and adjacent area is currently used for grazing, recreation and coal mining. Plates ~~2-2~~ 1-2 and ~~2-3~~ 1-3 coupled with Table ~~2-1~~ 1-3 show the fee ownership and leasehold interests adjacent to the permit boundary and the fee ownership of contiguous areas. This information provides a guide to the land uses of the various parcels.

1 **Surface Land Status/Mine Plan**

The land within the Bear Canyon Mine permit area fall under the jurisdiction of the State of Utah, U.S Forest Service, Emery County, and private surface owners.

County zoning ordinances classify the permit area as MG-1 (Mining and Grazing) and CE-1 (Critical Environment) as shown on Plate ~~2-2~~ 1-2. Site Plan approval has been issued by Emery County to approve mining.

Ownership

Plates ~~2-2~~ 1-2 and ~~2-3~~ 1-3 show the ownership of property within and contiguous to the permit boundaries. Ownership of land parcels within the permit boundaries are designated by capital letters. See Chapter ~~2~~ 1, Table ~~2-1~~ 1-3, for letter designation.

Surface Managing Authorities

Plate ~~2-2~~ 1-2 shows the surface ownership for each parcel within the permit boundaries. The local, state, and federal managing authorities are Emery County, State of Utah, Bureau of Land Management and the U.S. Forest Service.

1Utility Corridors and Other Right-of-Ways

Co-Op has been granted a mine access right-of-way in Section 26 along the Paved County Road accessing Bear Canyon. Utility corridors, such as power lines, telephone lines and water pipes, are shown on Plates ~~2-4~~ 5-2. All coal mining and reclamation operations will be conducted in a manner which minimizes damage, destruction, or disruption of services provided by oil, gas, and water wells; oil, gas, and coal slurry pipelines, railroads, electric and telephone lines; and water sewage lines which pass over, under, or through the permit area.

No oil, gas, or water wells, pipelines, railroads, electric and telephone lines, or water and sewer lines exist within the permit area except those associated with the mining operation, as described in ~~chapter 3~~ R645-301-521 and shown on plates ~~2-4A~~ 5-2A thru ~~2-4G~~ 5-2G.

Special use Permits and Leases

Co-Op leases land owned and leased by COP Development Company. Special use permits and information are shown in ~~Chapter 2~~ 1.

<u>Mineral Ownership/Mine Plan Area</u>

Other than coal, no minerals of value have been mined within the lease and permit area. No other mineral resources are known to be present in commercial quantities however there is potential for discoveries.

Coal Ownership and Mines (Permit Area and Contiguous Areas)

Coal ownership and mines in the permit area and contiguous areas are shown on Plate ~~2-3~~ 1-3 and listed with addresses in ~~Section 2.2~~ R645-34-112.330, Table ~~2-1~~ 1-1.

Coal Leases

The following coal leases are held by Co-Op adjacent to the permit area. For the locations of these coal leases, please refer to Plate 2-3.

Trail Canyon Permit Area
Bear Canyon Permit Area
BLM U-024316
BLM U-024318
BLM U-46484
BLM U-020668
BLM U-38727
BLM U-61048
BLM U-61049

Mineral Leases

BLM U-024316	(See Appendix ~~2-F~~ 1-E).
BLM U-024318	(See Appendix ~~2-F~~ 1-E).
BLM U-46484	
BLM U-020668	
BLM U-38727	
BLM U-61048	
BLM U-61049	

Oil and Gas Ownership and Leases

Oil and Gas Ownership and Leases

Co-Op represents no interest in oil or gas leases in the permit area. Federal Oil and Gas Leases U-38968 and U-58422 lie within the same area.

R645-301-411 Environmental Description

HISTORICAL AND CULTURAL RESOURCES

SCOPE

The project area is situated in the Wasatch Plateau mining area, approximately 15 miles west-southwest of Huntington, Utah USGS 7.5 Minute topographic quads of the project area include those and adjacent areas.

Surface within the large intensive survey area include privately owned, state, and Bureau of Land Management (BLM) administered lands, and U.S. Forest Service.

Environment and Locality

The Co-op Mining Company (Co-Op) project area is located on the east flank of the Wasatch Mountain Range. The highland locations are situated above the 8,000-ft. elevation adjacent to the and within Manti-LaSal National Forest while the larger mine facility in Bear Canyon lies at the base of the Wasatch Plateau between the 6,800 and 8,000-ft. elevations.

The survey area contains a wide variation of associated vegetation communities because of variations in soil, slope, elevation, and subsurface moisture retention. The rolling ridges and arroyos of the survey units incorporate pinyon and juniper communities within the broken arroyo drainage system; there plants gradually reduce their dominance upon the high flats where sagebrush vegetation exists. Serviceberry, rabbitbrush, and scattered saltbush plants also exist along these drainages. The steeper areas ascending the plateau contain mountain shrub communities, which include live oak and mountain mahogany. North-facing slopes contain Douglas fir in the drainages above the upper juniper zone.

411.100 Pre-Mining Land Use Information

History of Land Use

Prior to the beginning of the Holocene Epoch (about 10,000 years ago), the pluvial conditions of the Pleistocene in the eastern Great Basin and in the Wasatch Range began to decrease. The gradual heating and drying trend of the Anathermal (about 10,000 to 7,500 years ago) was accelerated until about 4,000 years ago, although this occurrence varied in different localities throughout the West relative to local conditions. The ecosystems of the project area were influenced by these climatic changes from cool and wet through a period of increasing desiccation. About 4,000 years ago, the climate in the Intermountain West became cooler and wetter than at present with a subsequent remigration of floral and faunal species from the upper elevations back into the lower basins. These fluctuations in climate affected prehistoric human occupation patterns in the West, as shall be noted in a later section.

Land use techniques employed in the project area have ranged from hunting-gathering activities, which began during the pleistocene, to primitive farming technology practiced along the river bottoms by the Fremont peoples as early as 1,500 years ago. With the introduction of the Euro-American settlers in the 19th century, modern farming technology, including horticulture and livestock production, became established in the Castle Valley area. From the historic period to the present, the general project area has been primarily utilized as livestock grazing land. Some horticulture related to the livestock industry has developed along the alluvial creek bottoms that extend to the east along the drainages. In addition, some coal mining has occurred during the 19th and 20th centuries at the Wattis mines to the North and at the site of the existing mine.

The South East Utah coal region encompasses lands in general, state, county, and private ownership. Land use management plans for public and National Forest Lands generally allow for mine and mine-related activities.

Coal mining has been an integral part of the region's economy. Mining and related construction activity dominate employment in Emery county. Active mining is going on in areas adjacent to the project area.

Historically, the livestock industry has been an integral part of the region's economy. Early settlers depended on range land for grazing sheep, cattle and horses. As time passed, grazing operations became smaller, more numerous, and directly associated with small farms. Timber also has been tied to an integral part of the economy of the region, but on a much smaller scale than the livestock industry. Early settlers needed fence posts, corral poles, house logs, mine timber, railroad ties and lumber; numerous small sawmills supplied local needs. As time passed and needs diminished, most mills went out of business. Recently however, commercial timbering has begun to increase in the region.

411.110 Use of Land Existing at Time of Filing

The uses of the land at the time of filing of the permit application were coal mining, wildlife habitat, livestock grazing and outdoor recreation.

411.120 Capability of Land to Support a Variety of Uses

Present land capability and productivity will be only slightly reduced compared to the after mining capability due to the small area of actual surface disturbance. Mining activities have proceeded on the current lease areas of Co-Op historically with only minor effects on productive capabilities in terms of soils, topography, vegetation or hydrology. The soils indigenous to the area affected by the operations are described in Chapter 2. Vegetation is discussed in Chapter 3.

Surface water in the permit area is limited to surface run-off that flows most heavily during the spring and early summer months and then normally dry up. The quality and quantity of this water and of the ground water will be identified in Chapter 7.

Land productivity in terms of plant products before any mining will not differ greatly from future productivity due to the small area of actual surface disturbance. Early settlers depended upon range land for grazing sheep, cattle and horses. Timber was active, but on a much smaller scale than grazing. Early settlers needed fence posts, house logs and railroad ties.

The permit area affected by surface operations and facilities of the underground Bear Canyon mine is capable of supporting grazing and recreational uses. Grazing is most probable within Leases U-024316 and U-38727. Farming in the area is prohibited by the steep and rocky terrain.

Current and future land use will suit the physical features of the mine plan area, which is mostly steep and rocky. Such land is well suited for management as multitude area and coal mining fits appropriately into the overall land use scheme.

Land productivity data were obtained from the U.S. Soil Conservation Service, and are included in Chapter 3.

Present management emphasized livestock and wildlife grazing, and watershed development. Coal preparation and management facilities are located on fee land.

Grazing. Private land owned by COP Development Company in and contiguous to the Permit area is presently used for grazing. Grazing occurs on Leases U-024316, U-020668 ~~and~~ , U-38727, U-61048, and U61049 is managed by the U.S. Forest Service.

Recreation. Recreational use of the area affected by mining operations consists primarily of hunting and camping. Heavy hunting of mule deer occurs on the area. Camping frequently occurs on land adjacent to the property. The property owned by C.O.P. Development Company in and adjacent to the permit is currently leased to Sportsman's, Inc. as part of a Private Hunting Unit (PHU). This PHU includes a hunting cabin located adjacent to the Wild Horse Ridge within the permit. Recreational access to this facility is provided by the Bear Canyon Haul Road, the Wild Horse Ridge #3 and #4 Mine access roads.

Forestry. Merchantable timber is found within the mine permit area primarily on the flanks and top of McCadden Ridge within Lease U-024316, although much of the area is covered by pinyon pine and juniper. Limited resources also exist in the bottom of Bear Canyon, primarily within the Right Fork area.

Mining. The type and extent of mining activities are discussed in detail in Chapter 5.

411.130 Land Use Classifications Under Local Law

The land within the Bear Canyon Mine permit area fall under the jurisdiction of the State of Utah, U.S Forest Service, Emery County, and private surface owners.

County zoning ordinances classify the permit area as MG-1 (Mining and Grazing) and CE-1 (Critical Environment) as shown on Plate 1-2. Site Plan approval has been issued by Emery County to approve mining.

411.140 Cultural and Historical Resources Information

The Division of State History was contacted in reference to that portion of ground in T16S, R7E, Sec. 23, 24 and 25 that has been or may be disturbed. It was the conclusion, in both conversations, that:

a. There are no known sites of any significance existing in the area in question.

b. That the majority of the land in question has been previously disturbed due to earlier mining activities

c. That a survey of areas of future disturbance may be advantageous but to survey ground which is disturbed serves no purpose.

However, in the event that C. W. Mining is in a position to permit new facilities on disturbed ground, it has committed to a thorough Paleo-Archo Survey prior to any new disturbances. Also, should any evidence of Pale-Archo finds be discovered in the course of present construction, the site will be roped off and construction halted until the Historical Division is contacted. However, a survey was conducted the summer of 1984 and 1990 for those areas which may be adversely impacted by subsidence. This information was submitted as Appendix 4-A. Appendix 4-B contains the results of a survey of the Wild Horse Ridge Area which was conducted in 1982. Appendix 4-C contains the results of a survey of the Wild Horse Ridge Tank Seam Area conducted in 2001. Appendix 4F contains the results of a 2004 cultural resource survey of the Wild horse Ridge subsidence area. Appendix 4H and 4I contain 2006 cultural resource studies for the Mohrland lease and fee area addition.

At the request of the U.S. Forest, an additional thorough literature search will be conducted for any cultural resources within those areas that may be adversely impacted by subsidence. Co-Op Mining Company commits to conducting this literature search prior to any retreat mining within the Wild Horse Ridge area.

Application of the National Register Criteria of Eligibility, as defined under 36 CFR 60.6, indicates that there is one site within the permit area which would be considered a candidate. This is the Bear Creek Rock Shelter (Site 42 EM 1572).

411.141Cultural and Historic Resources Maps

These maps are located inside the reports on the specific areas.

411.142 Cordination With State Historic Preservation Officer

During the permitting of the Bear Canyon Mine, Co-Op counseled with the Utah Division of State Historical Preservation Office and agreed to an on-site survey. The survey was conducted by John A. Senulis, an approved archaeologist (Senco-Phenix). The survey and results are included as Appendix 4-A. Co-Op is committed to take all necessary steps to protect any sites deemed necessary in the event any are located. Mr. Senulis also conducted a survey of the Federal Lease U-024316. The results of this survey can also be found in Appendix 4-A. Two surveys of the Wild Horse Ridge area have been conducted. The first was a survey by Kenneth Juell of the University of Utah Archaeological Center in 1982. This survey covered drill sites and access roads both on top of the ridges and in the canyon.

According to Beaver Creek Coal Company, the survey revealed (Site 42 EM 1572) and a single other historic resource. The historic resource (42 EM 1572) was excavated by Nielson & Schleisman in July, 1982. The report of the excavation is included in Appendix 4-B. The other historic resource was found on the ridge while moving from one sample section to another. It was not considered significant or diagnostic.

A survey was also conducted by Heather Weymouth of Sagebrush Consultants in 1999. The results of this survey is also included in Appendix 4-B. No additional cultural resources were identified.

411.143 Identification of Historic and Archeological Resources

See 411.140, 411.141, and 411.142.

Additionally C. W. Mining Company conducted a search for paleontologic date within the general area. The purpose of the search was to:

a. Identify all known paleontologic sites within the designated area.

b. Identify stratigraphic horizons which are potential producers of paleontologic resources.

c. Evaluate the uniqueness of known or potential fossil sites compared to similar or duplicate faunas from the same stratigraphic horizon in other nearby areas.

Most of the ground surface within the general area is composed of the Masuk Member of the Cretaceous Mancos Shale. The Masuk is the uppermost shale member of the marine Mancos, overlying the Emery Sandstone Member and underlying the Star Point Sandstone. The lithology of the Masuk is silt, mudstone, and shale. It is about 1,000 ft. (305 m) thick in the permit area, and covers most of the area in question. Above the Masuk Member is the Star

Point Sandstone, transitional marine_nonmarine sandstone bed which is approximately 500 ft. (152m) thick.

The marine Masuk Shale contains a widespread fauna consisting of abundant foraminifera (Maxfield, 1976). Ammonoids, bivalves, gastropods, fish and turtle teeth (Fisher, 1960), and probably also ostracodes (Lessard, 1973).

The Star Point Sandstone, a deltaic sequence, has produced only trace fossils from the general area. Burrowing remains of two generic types have been described from the Star Point Sandstone by Howard (1972) and Marley et al. (1979).

In every case these fossils have been reported over broad areas surrounding the study site; therefore, it is almost certain that the Masuk Shale and the Star Point Sandstone within the study area contain similar fossils.

Although no specific paleontologic sites within the designated study area are reported in published literature, there are many occurrences in the surrounding areas, strongly indicating the presence of these same fossils at the study site.

Previous paleontologic investigations demonstrate widespread occurrences of the faunas within Late Cretaceous marine and nonmarine strata in this area. Therefore, all fossils which likely occur within the study site are almost certainly duplicated in surrounding outcrops of Masuk Shale and Star Point Sandstone.

The Paleontologic resources within the permit area are neither particularly abundant nor unique compared to their counterparts in similar stratigraphic horizon within the general area. Based upon present knowledge, development of this site would not pose a threat to the paleontologic resources of the area.

411.144 Protection of Historic and Archeological Properties

A variety of archaeological and historic techniques are available for use in avoiding and protecting sites, or for mitigating potential adverse affect to significant cultural resources. Such actions, once proposed, are contingent upon comments from relevant Department of Interior agencies and Utah State Preservation offices. Avoidance procedures are the most effective means of preserving cultural resources and will be implemented in the event that a site is uncovered.

411.200 Previous Mining Activity

The permit area in was the site of an active coal mine. There is very little information available concerning the history of previous mining activity in Bear Canyon. The following information was taken from, Central Utah Coal Fields, Utah Geological & Mineralogical Survey, 1972 and from local people who were in the area back to the 1940's.

Bear Canyon enjoyed two periods of activity between 1885 and 1906, during which the coal seams were worked spasmodically. The Bear Creek Mine was owned and operated by Sam Holl (Mcelpay, 1949) and later by George A.S.R. Two seams were mined; the Bear Canyon Seam and the Hiawatha Seam. After 1906, the mine operated steadily and continuously. The twelve prospects in the canyon produced about 8,000 tons in 1906. Coal was removed with pick and shovel. Mining areas were dictated by the locations where seams were exposed.

Beginning in 1906, the land was transferred form George A. Smith to the Freed family, and in 1931 to Freed Coal and Coke. In 1943, the land was transferred to Karsen Co., and then back to Freed in 1946.

During this time, the mine was operated by a man named Stobaugh. Stobaugh used powder to loosen the coal and horses to haul the coal out of the mine. Most of the mining was done by hand. The initial area with erratic haulways shown on Plate 5-1A, was mined during this period. R. McCandless and S. McCarther operated the mine up until 1957, using similar methods. The area mined during this period is also shown on Plate 5-1A with a more regular pattern. From 1938 to1957 the mine produced over 150,000 tons from the Bear Canyon Mine. Co-Op entered the Blind Canyon Seam through these workings.

In 1957, the land was transferred from Freed to Huntington Corp., and then Peabody Coal in 1971. From 1971 to 1977, various exchanges took place between Peabody and Nevada Electric Investment. In 1980, the original Co-Op lease area was transferred from Peabody to C.O.P. Development. In 1990 the Wild Horse Ridge was transfered from Nevada Electric to COPD. In December 1996 the Mohrland area was transferred from Intermountain Power Agency to C.O.P Development

R645-301-412 Reclamation Plan

412.100 Post-Mining Land Use Plan

Table 4-1 Proposed Post-Mining Land Use

Land Use in Relation to Mine Features

Area	Present Ownership	Pre-mining Use	Proposed Post-mining Use	Alternate Use
Mine Site Exploratory Excavations	Private	Wildlife/ Grazing/ Recreation	Wildlife/ Grazing/ Recreation	Picnic Area
Conveyor, Pipeline and Power Line Route	Private	Grazing	Grazing	Wildlife Habitat
Main Access	Private	Service Road	Service Road	Wildlife Habitat
Tank Seam Access	Private	Wildlife	Wildlife	
Wild Horse Ridge	Access/ Conveyor and Private	Wildlife/ Recreation	Wildlife/ Recreation	Timbering

Land Use in Relation to Physical Features

Area	Proposed Post-mining Use	Ability to Support Proposed Post-mining Use
Flatlands	Wildlife/Grazing Habitat/ Timber/Recreation	Adequate
Canyons	Wildlife/Grazing Habitat/ Recreation	Adequate
Moderate Elevation: North & East Slopes	Wildlife/Grazing Habitat Recreation	Adequate
High Elevation: Steep land North & East Slopes	Wildlife Habitat	Adequate
West and East Slopes	Wildlife Habitat	Moderate - Because of Harsh Natural Conditions

412.110 Method For Achieving Post-Mining Land Use

R645-301-540 describes in detail, the abandonment steps and revegetation/reclamation activities to be used to achieve the proposed post-mining land uses.

Area Cleanup. Solid waste generated in the abandonment operation will be disposed as described in R645-301-541.300.

Recovering of the General Area. Grading and backfilling will be done to achieve a final contour suitable for the wildlife/grazing/recreation habitat specified as the post-mining land use.

Wind Protection Barriers. In addition to the wind protection provided by the soil stabilation, rock wind barriers may be constructed by a small portion of the rock generated during the mining operation. During abandonment small piles of the rock may be formed where needed to provide protection and stability to reclaimed areas.

Scarifying Areas. Operational areas will be scarified after backfilling and grading prior to topsoil redistribution. Steep slope areas which must remain after abandonment will receive special ripping to create ledges, crevices, pockets and screes. This will allow better soil retention and vegetation establishment.

Distribution of Topsoil. Topsoil from the stockpile will be spread over the disturbed areas in such a manner as to prevent excessive compaction.

Fertilization and Neutralization. Fertilization or neutralization as determined ~~as~~ necessary by soil testing will be done.

Seeding and Tree Planting. Vegetation will be established to prevent erosion, to optimize the effect and to provide cover. Perennial woody species will be emphasized, along with those of proven nutritional value and ability to support wildlife and grazing. The types and amounts of such vegetation are discussed in Chapter 3 R645-301-321.

Moisture Retention. All regraded and topsoiled areas will be mulched or otherwise treated to promote germination of seeds and to retain moisture. Various other methods available are listed below:

1. Straw--Terrace Benches
2. Mulch--Wood mulch may be sprayed on terrace banks
3. Soil Retention Blanket--Wood fiber held by plastic net may be used on steeper banks.
4. Jute Mesh and Straw--Burlap material holding straw may be used on the steepest banks.
5. Tackifier--Mulch with tackifying agent may be used on steep banks.

Maintenance. Fencing, irrigation and weed control will be used only as needed, according to operational testing results.

Regrading and Reseeding. Erosion that develops in completed areas will be minimized by repeated grading and seeding.

Success Monitoring and Extended Responsibility Period. Vegetation and water will be monitored during the applicable period of liability to determine success of abandonment reclamation. A determination of revegetation success will then be made.

412.120 N/A

412.130 Alternative Post-Mining Land Use

No proposed post-mining land use is different from the pre-mining land use.

412.140 Considerations of Surface Owners

Consideration has been given to make all of the proposed land use consistent with owner plan and applicable Utah and local land use plans. Additionally correspondence will continue between C. W. Mining Company and the land owners to assure proposed post mining land use will be consistent with their land use plans. Copies of this correspondence will be included in Appendix 4-C.

412.200 Land Owners Comments

This is included in Appendix 4-C.

412.300 Suitability and Compatiblity

Plans for final fills and surface regarding is discussed in Chapter 5.

R645-301-420 Air Quality

R645-301-421 Compliance with the Clean Air Act

Coal mining and reclamation operations are conducted in compliance with the Clean Air Act and the Utah State Department of Health Air Conservation Regulations as outlined in the Air Quality Approval Order found in Appendix ~~4-D~~ 4-G.

R645-301-422 Coordination and Compliance with Utah Bureau of Air Quality

C. W. Mining Company is operating under Approval Order # DAQE-145-02 (Appendix ~~4-D~~ 4-G) issued by the State of Utah Department of Environmental Quality Division of Air Quality. Additionally C. W. Mining Company is regularly inspected by representative of the Division of Air Quality to assure compliance with the Approval Order and the Clean Air Act and the Utah State Department of Health Air Conservation Regulations .

R645-301-423 Air Pollution Control Plan

C. W. Mining Company will control air pollution by watering roads, drop points, and storage piles as outlined in Approval Order # DAQE-145-02 found in Appendix ~~4-D~~ 4-G.

Land Use

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Appendix 4-A

PALEO-ARCHEOLOGICAL SURVEY

This information was relocated to the confidential binder on 9/18/05.

Confidential files are located at the Division of Oil Gas and Mining 1594 West, North Temple, SLC, Utah.

Appendix 4-B

WHR Resource Survey

This information was relocated to the confidential binder on 9/18/05.

Confidential files are located at the Division of Oil Gas and Mining 1594 West, North Temple, SLC, Utah.

Appendix 4-C

Property Owners Statement

C.O.P. COAL DEVELOPMENT COMPANY
53 West Angelo Avenue
Salt Lake City, Utah 84115

December 14, 1990

State of Utah
Division of Oil, Gas & Mining
355 West North Temple
3 Triad Center, Suite 350
Salt Lake City, Utah 84180-1203

Gentlemen:

Please be advised that the pre-mining use of the real property now being used for mining operations by Co-op Mining Company at its Bear Canyon Mine, was for wildlife, recreation and livestock grazing. C.O.P. Coal Development Co., as the land owner, presently anticipates that the land will be used for coal mining activities for the next several years, after which time, the land will once again be used for wild life, recreation and livestock grazing.

Very truly yours,

Joseph O. Kingston

Joseph O. Kingston, President
C.O.P. Coal Development Co.

JOK/kj

Appendix 4-D

Air Quality Monitors

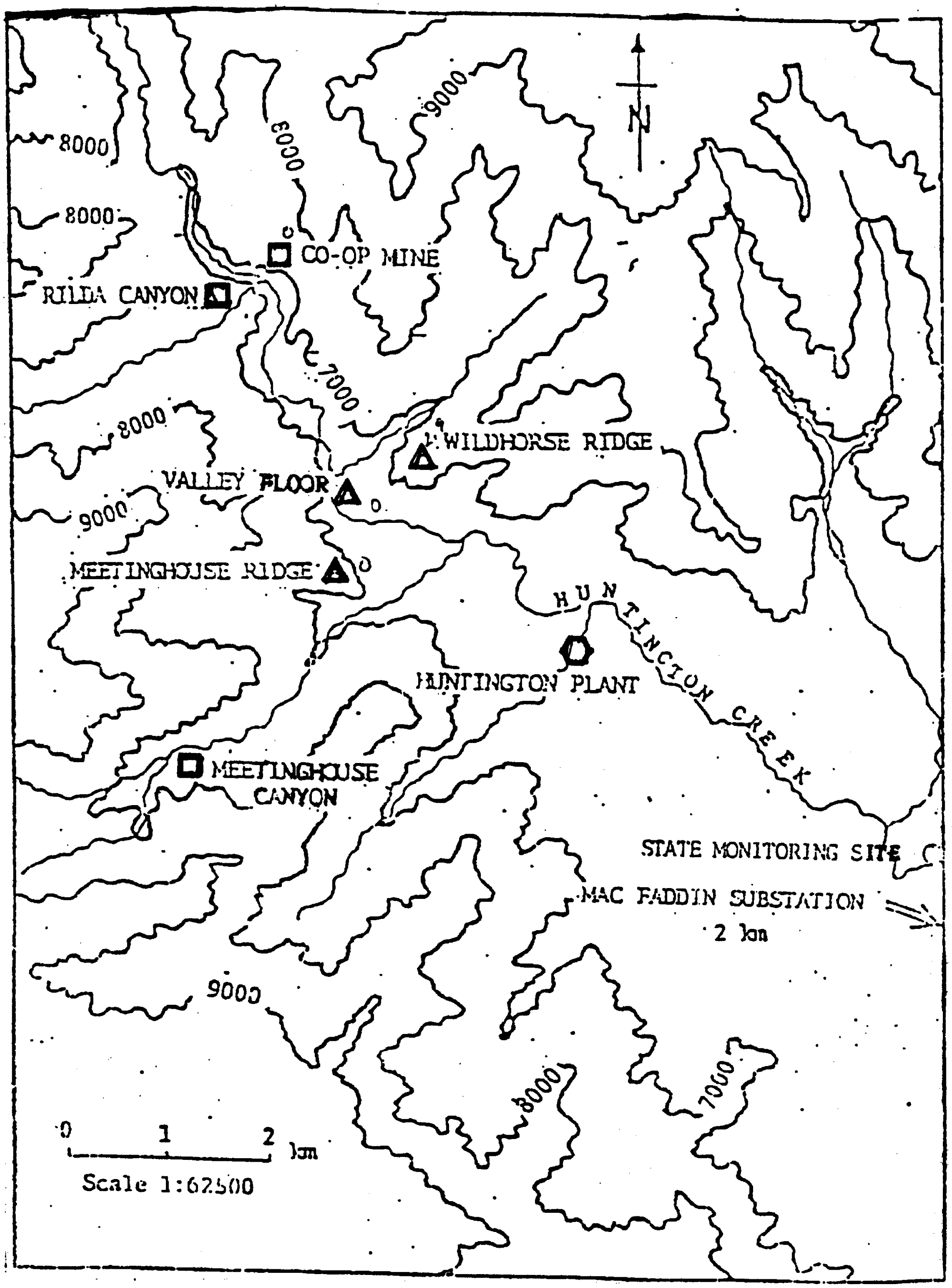


Figure 4D-1 Air Quality Monitor Locations, Huntington Canyon, Emery County, Utah

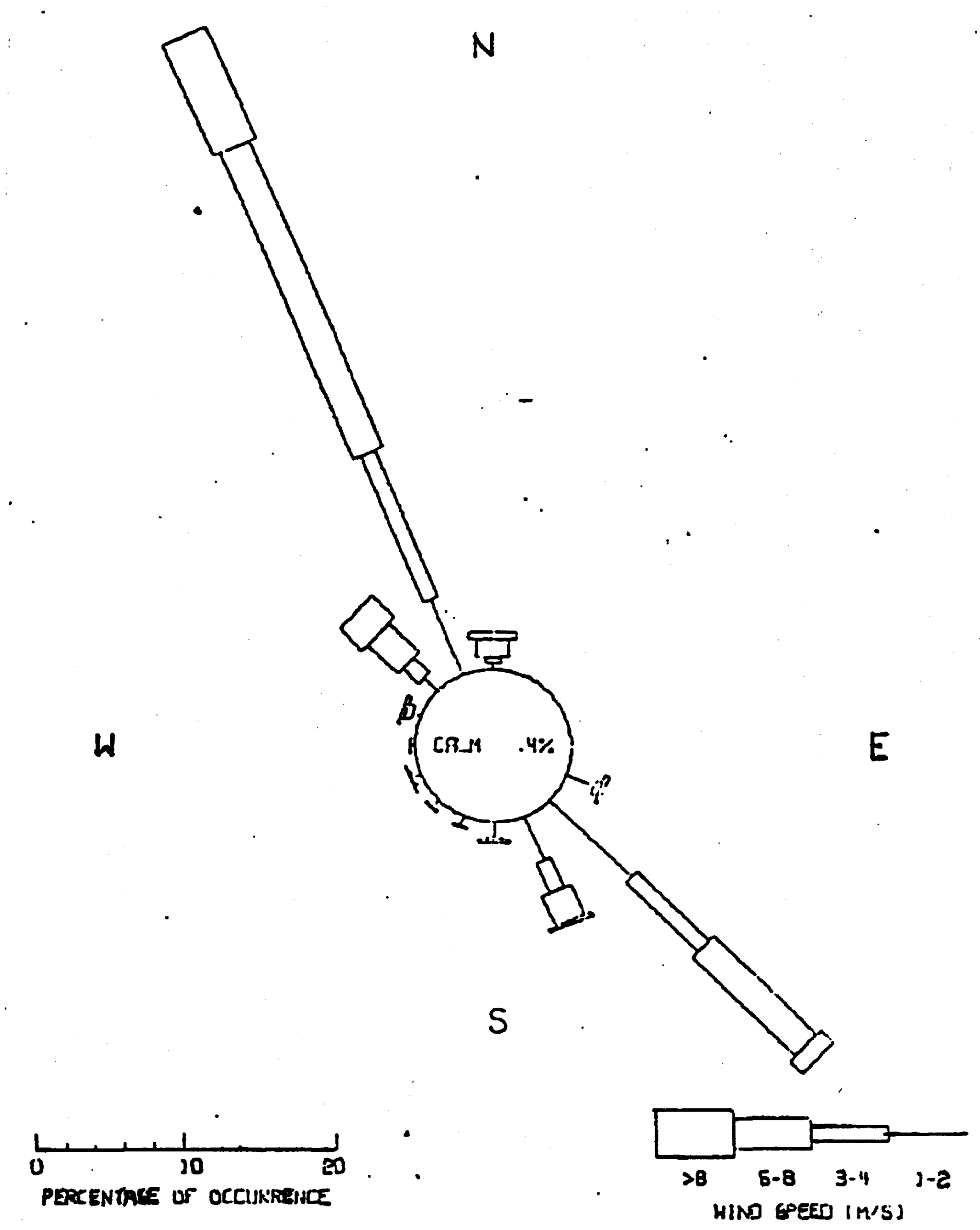


Figure 4D-2 Wild Horse Ridge, October 1977 thru April 1978, All times.

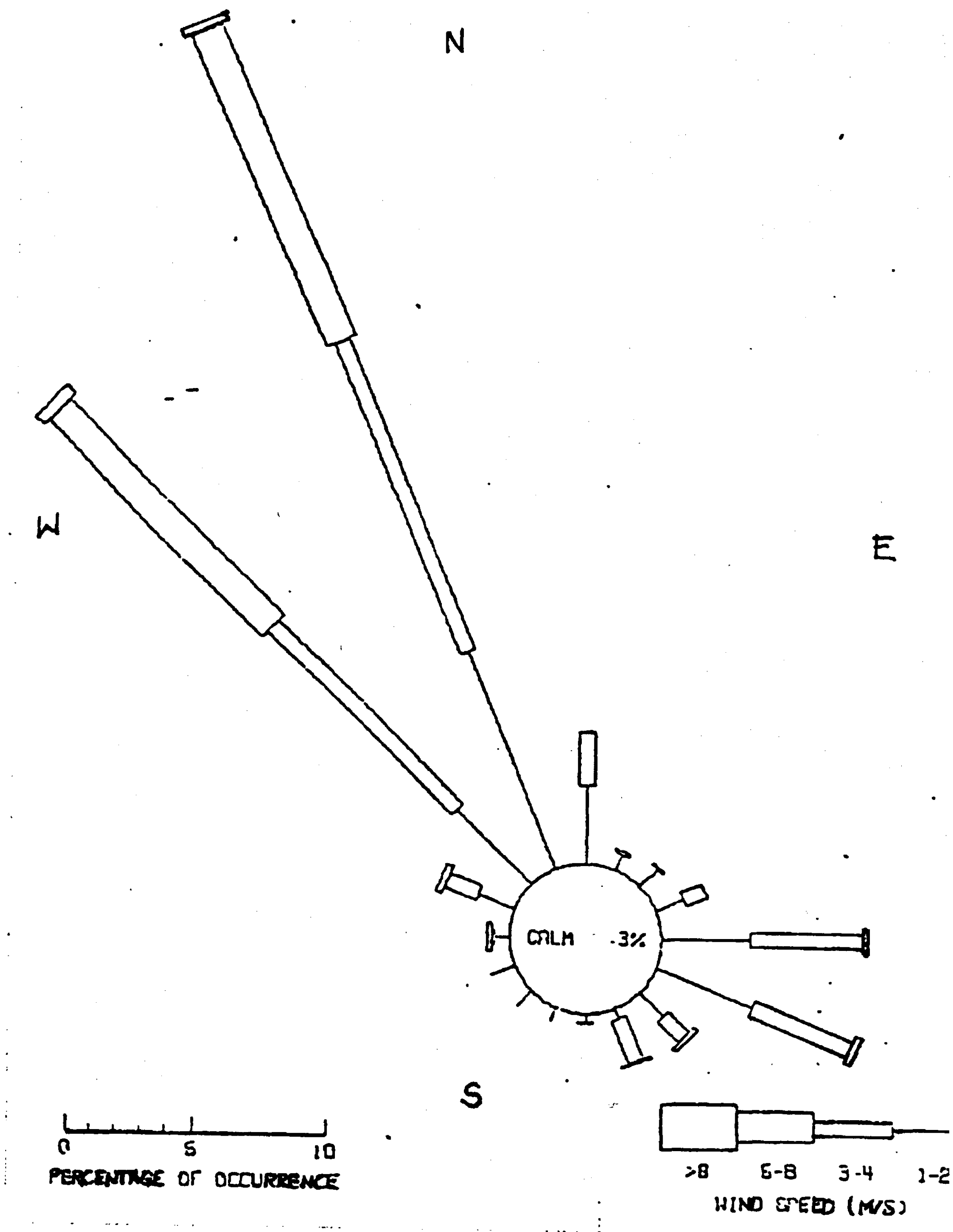


Figure 4D-3 Valley Floor, October 1977 thru April 1978, All Times.

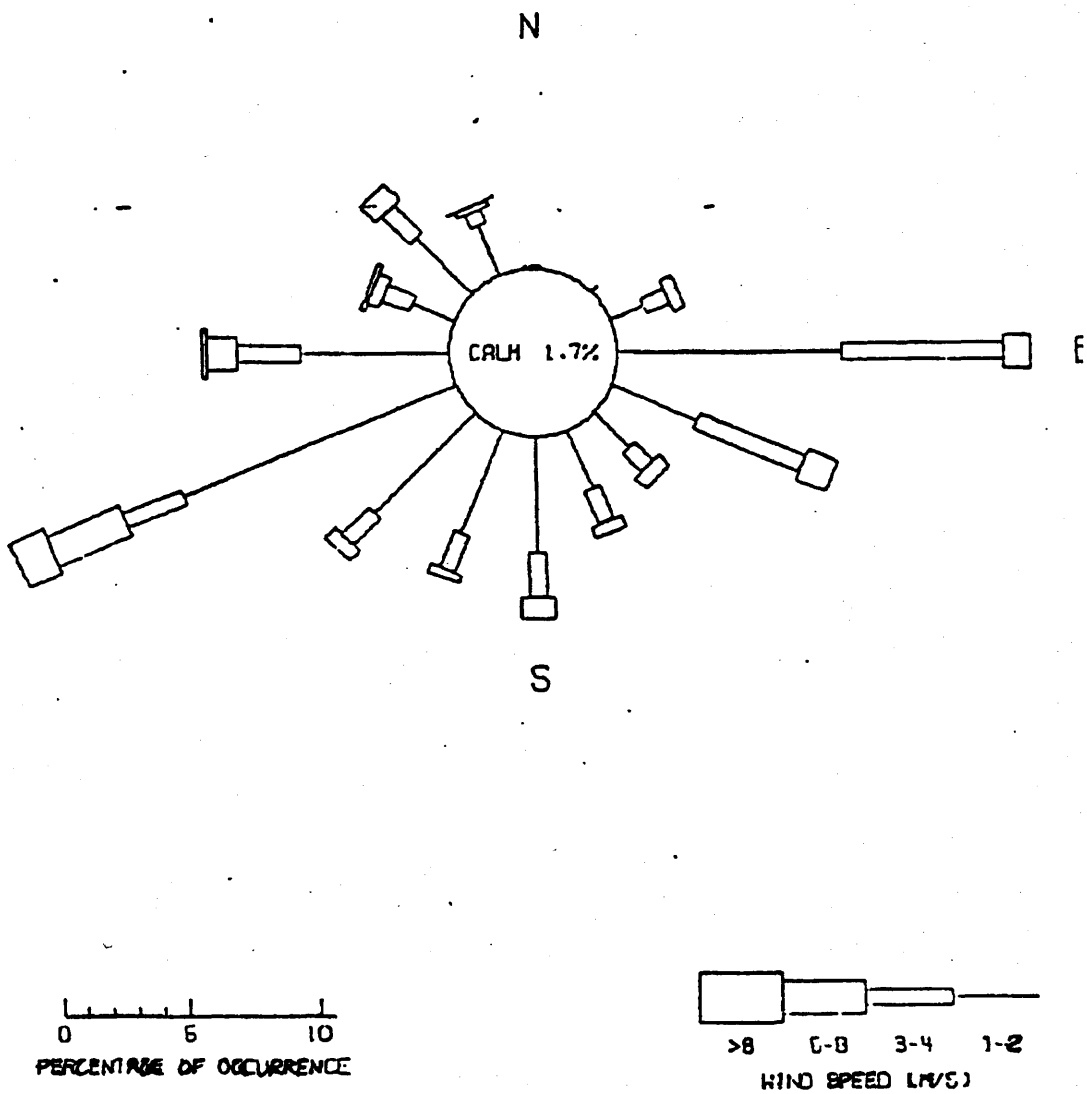


Figure 4D-4 Meeting House Ridge, October 1977 thru April 1978, All times.

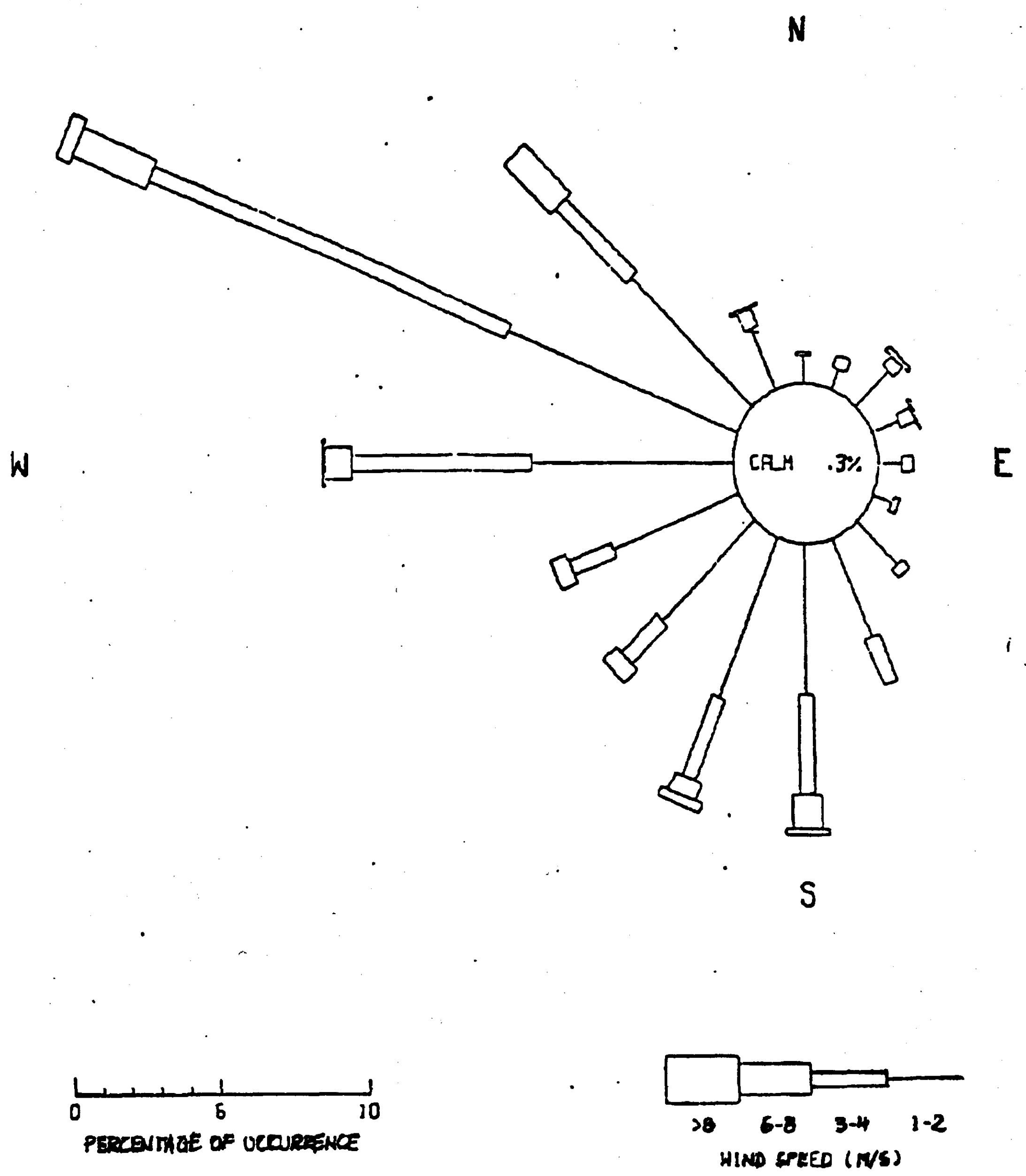


Figure 4D-5 Mac Faddin Station. December 1977 thru April 1978, All times.

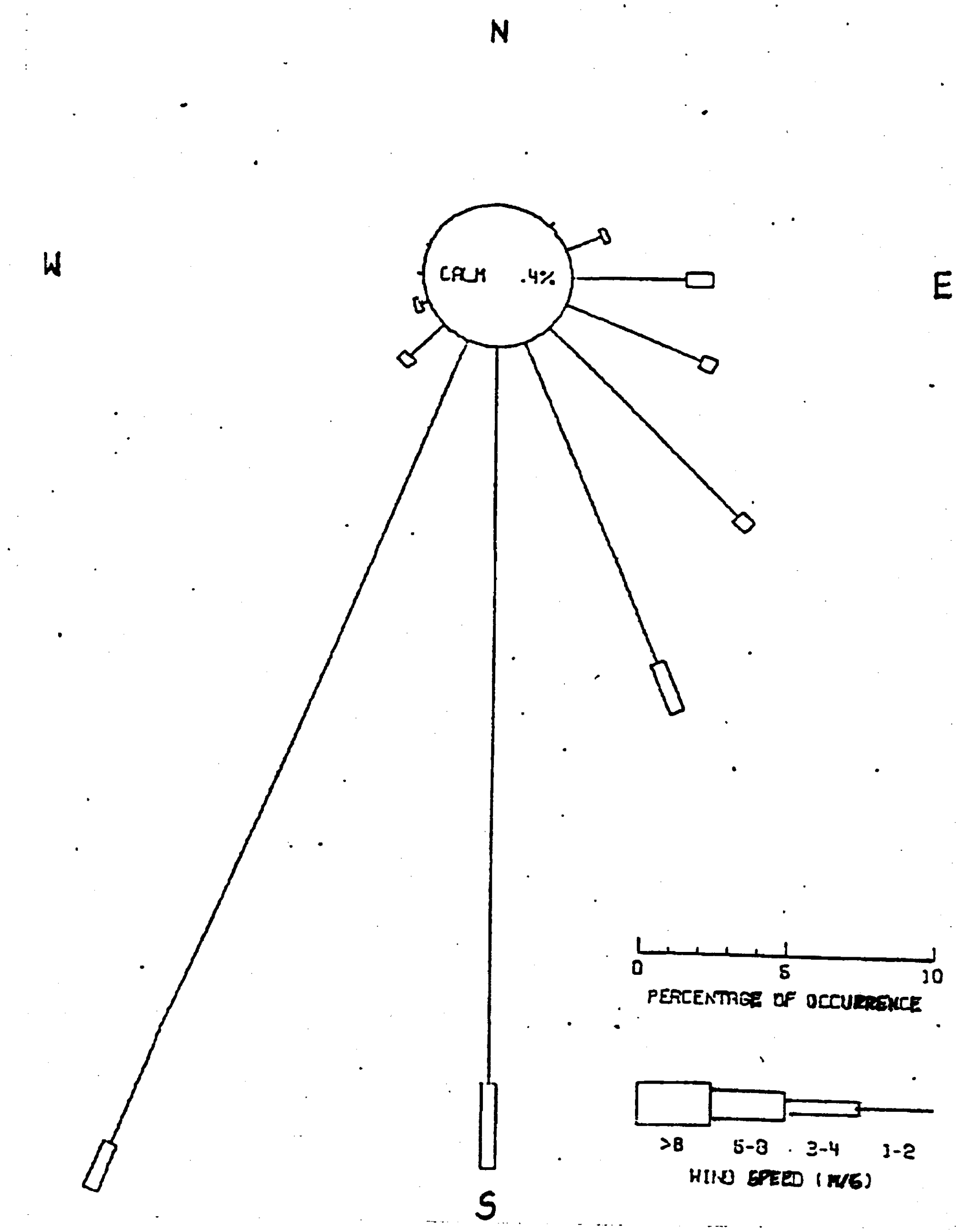


Figure 4D-6 Meeting House Canyon, December 1977 thru April 1978, All times.

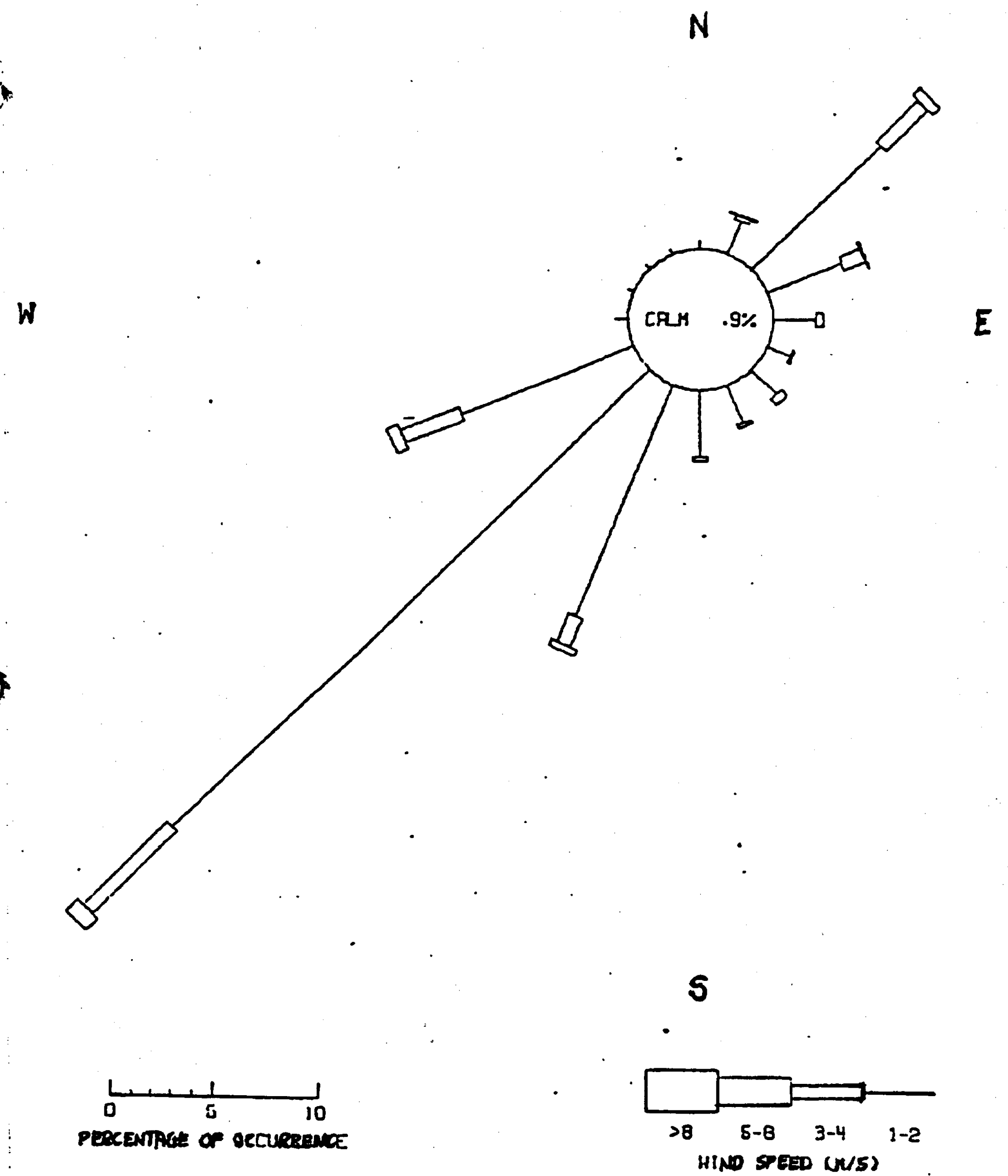


Figure 4D-7 Rilda Canyon, December 1977 thru April 1978, All times.

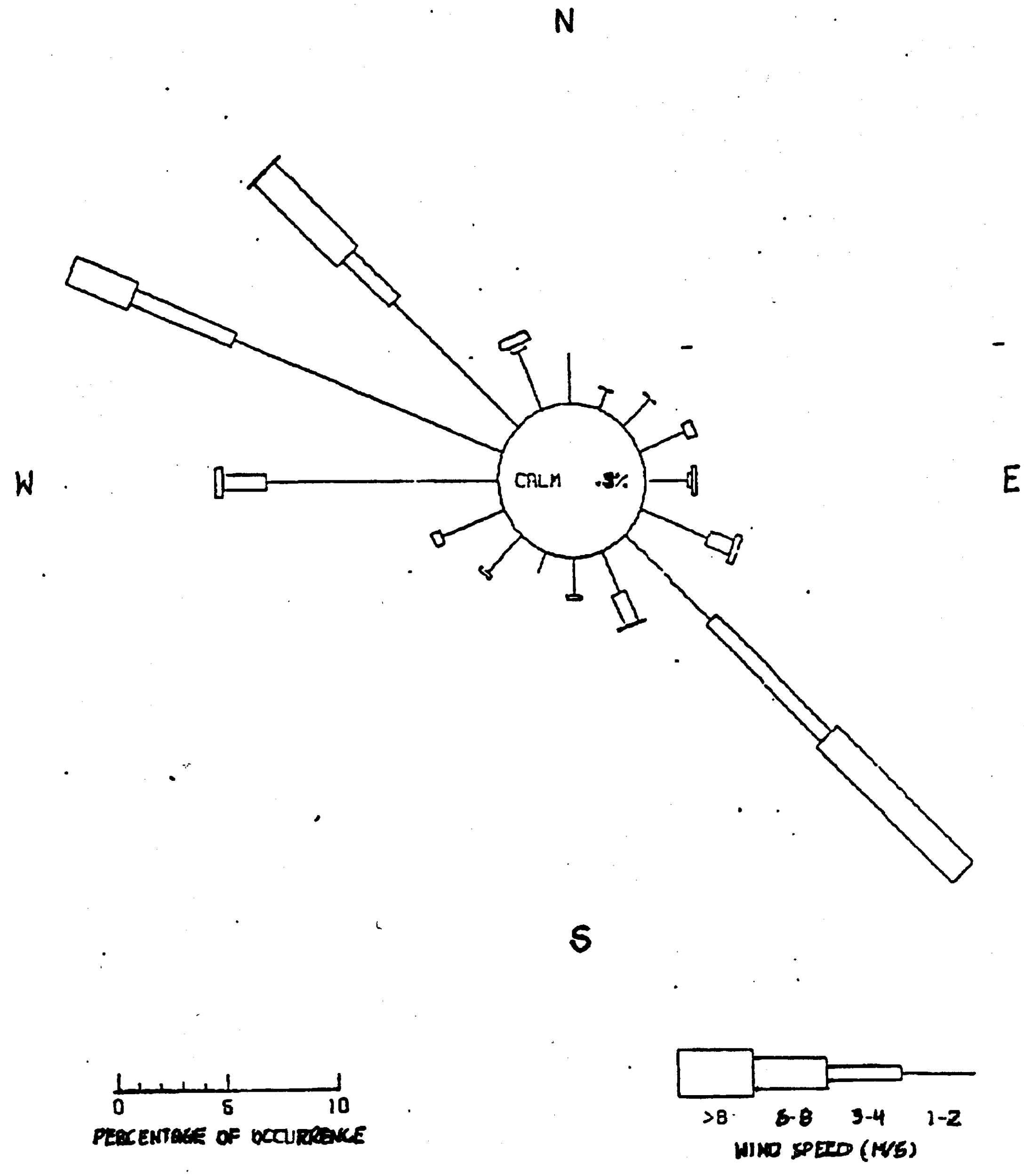


Figure 4D-8 Co-Op Mine, October 1977 thru April 1978, All times.

Appendix 4-E

Wild Horse Ridge Tank Seam Resource Survey

This information was relocated to the confidential binder on 9/18/05.

Confidential files are located at the Division of Oil Gas and Mining 1594 West, North Temple, SLC, Utah.

Appendix 4F
Cultural Resource Study of the Bear Canyon Subsidence Area

SENCO-PHENIX

AN INTENSIVE AND INTUITIVE CULTURAL RESOURCE SURVEY AND INVENTORY OF THE BEAR CANYON SUBSIDENCE AREA

Price Ranger District
Manti - La Sal National Forest

Emery County, Utah

PERFORMED FOR
C W Mining Company

In Accordance with Forest Service and
Utah State Guidelines
Antiquities Permit #U03SC0602f

SPUT-459
February 19, 2004

John A. Senulis

Direct Charge of Fieldwork

ARCHEOLOGICAL CONSULTING SERVICES • 1405 West 620 North • Price, Utah 84501 • PHONE 435.613.9091 • FAX 435.613.9116

UTAH SHPO

COVER SHEET

Project Name: **AN INTENSIVE AND INTUITIVE CULTURAL RESOURCE SURVEY AND INVENTORY OF THE BEAR CANYON SUBSIDENCE AREA**

C. W. Mining Company

State #U03SC0602f

Report Date: February 19, 2004 County (ies): Emery

Principal Investigator/ Field Supervisor: John A. Senulis/John Senulis

Records Search/Location/Dates: July 23, 2003, Price River Field Office of the BLM

Acreage Surveyed: 21 acres

Intensive Acres: 10 Recon/Intuitive Acres: 11

U.S.G.S. 7.5 Quad: Hiawatha, Utah (1978)

Sites Reported	Number	Smithsonian Site #(s):
Archeological Sites:	0	
Revisit (No IMACS update)	0	
Revisit (IMACS update attch.)	0	
New Sites (IMACS attached)	0	
Archeological Site Total:	0	
Historic Structures:		
(USHS Site Form Attached)		
Total NRHP Eligible Sites,	0	

Checklist of Required Items:

1. X 1 Copy of Final Report
2. X Copy of U.S.G.S. 7.5' map showing surveyed/excavated area
3. Completed IMACS Site Inventory Forms Including
   - ______Parts A and B or C
   - ______IMACS Encoding Form
   - ______Site Sketch Map
   - ______Photographs
   - ______Copy of USGS 7.5' Quad with Smithsonian site Number
4. X Completed Cover Sheet

CULTURAL RESOURCE SUMMARY FORM **USFS# ML-00**
MANTI-LA SAL NATIONAL FOREST USHPO# U02SC0205f
(Attach Narrative Report Form)

FS PROJECT NO.: ML- _ - _ Name **AN INTENSIVE AND INTUITIVE CULTURAL RESOURCE SURVEY AND INVENTORY OF THE BEAR CANYON SUBSIDENCE AREA**

Manti-La Sal	Price	T.16S, R. 8E Sec.19, 30
Forest Name	**District**	**T/R/Sec(s)**

February 19, 2004 SENCO-PHENIX /John Senulis)
Report Date **Organization/PI**
Benefitting Function:

__Heritage Project x__106 Compliance Project __Assessment or Resource Assistance (Not a 106 undertaking)

Project Cost: Vehicle/Gas NA USFS Amt. $ 0
Supplies Proponent Amt. $
Salary 0 Contributed Amt. $

(2xvol) Per Diem/Vol______TOTAL $ 0
Other______
Overhead

Nature of Work (Check all that apply):
x Survey
__ Monitoring /Docum.
__ Evaluation or Test Excavation
__ Interpretation
__ Site Protection/ Stabilization
__ Other (Explained below)
__ MOU/Clearance (No new surv)
__ Assessment/Plan
__ Mitigation: Excavation or Documentation
__ Discovery
__ DamageAssessment/ Vandalism

Total Project Acres: 21
Acres Surveyed: 10
New Sites Recorded: 0
Sites Monitored W/ Documentation: 0

Project Photographs Catalog #s:
Artifacts (isolates) Collection #s: None
Location of Curation: N/A

Tracking Dates (N/A if not required):
To SHPO___ __
Concurrence Rec'd_________
Mylar Updated_________
GIS Digitized_________
Proj. Database Entry_________
Accomplishment Report :

_ Not a 106 Undertaking/No ground disturbance
X No Effect; No Sites
No Effect Through Design or Mitigation Rec's
_ Other Effect (See Detail Report)
_ No Effect, No Eligible Sites

No sites located

Determination of Effects on Sites

Certification:
John A Senulis
Reporter's Signature Date 02/19/2004 Reviewer Signature Date

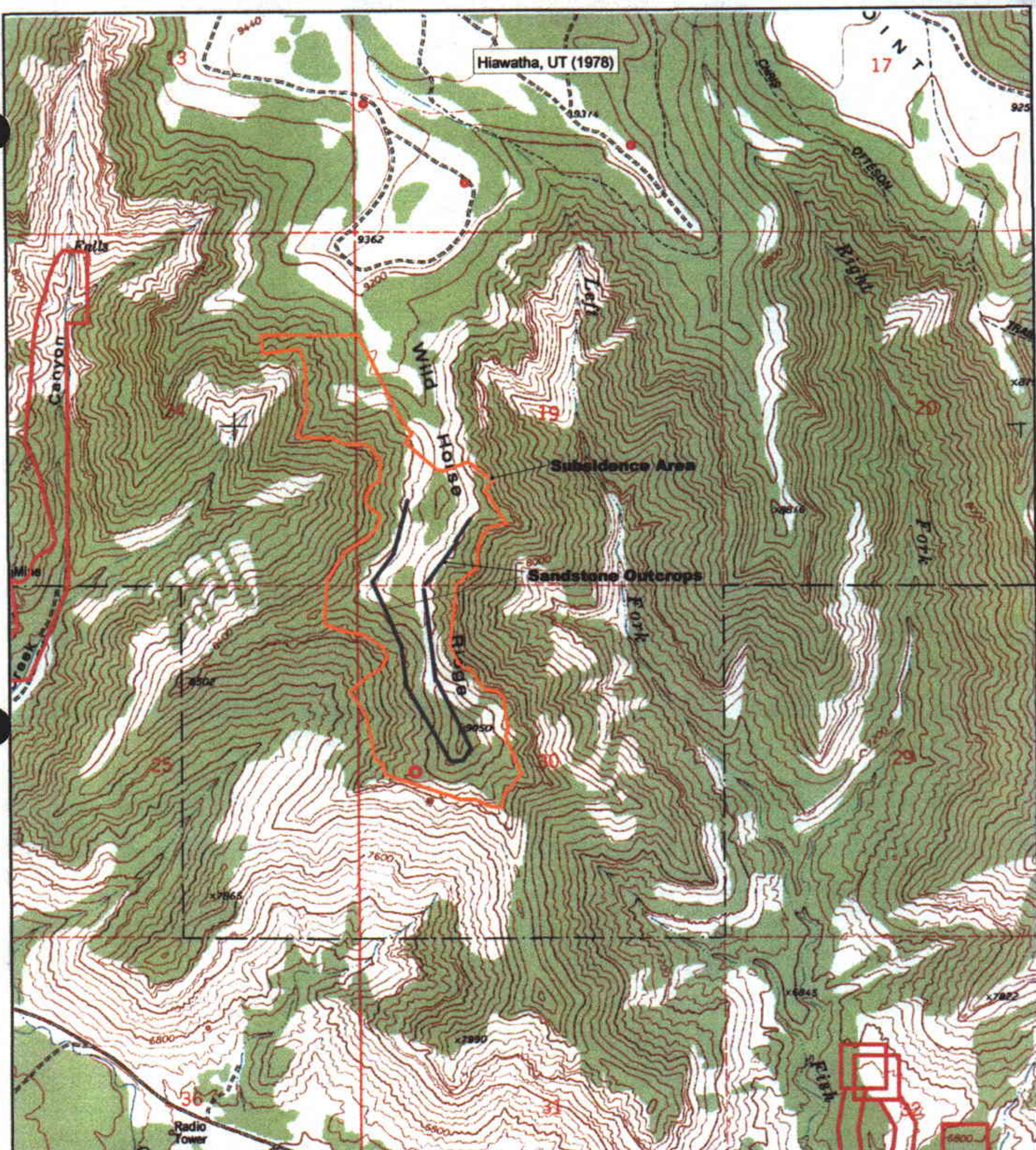
COMMENTS:

FSM 2360 MLNF/4-98

Abstract

SENCO-PHENIX performed a combination intensive and intuitive cultural resource survey of the sandstone cliff faces within the proposed Bear Canyon Mine subsidence area for C W Mining Company. The project area was the sandstone cliff faces that may subside when the pillars for the underlying Bear Canyon mine are removed. The focus was on cliff faces because during subsidence, cliff faces tend to shear off and collapse while there is a minimal effect on other ground. The subsidence survey was undertaken at the request of Forest Archeologist, Bruce Ellis, who wanted the areas of the cliff faces examined for possible archeological remains such as rock art, rock shelters, burials or other cliff face type sites. The project area is in the Price Ranger District of the Manti-La Sal National Forest. The purpose of the survey was to identify and evaluate cultural resources that may exist within the project area.

No cultural resources were located and the potential for undetected remains is remote. A finding of no effect is appropriate and archeological clearance without stipulations is recommended.



N

Scale 1:24,000
1"= 2,000'

Current Survey

Previous Survey

Eligible Sites

Ineligible Sites

Subsidence Survey
C. W. Mining Company
Emery County, Utah
Sections 19, 30, T16S, R8E
February, 2004
SPUT-459

SENCO-PHENIX

Project Location

The survey area is Wild Horse Ridge, which is the divide between Fish Creek to the east and Bear Creek to the west. Both are branches of the Huntington Creek drainage within the Price Ranger District of the Manti-LaSal National Forest in Emery County, Utah. The subsidence area is in Sections 19 and 30, T16S, R8E, Emery County, Utah. The project area is shown on the enclosed copy of U.S.G.S. 7.5' Quad: Hiawatha, Utah (1978).

Environment

The project area is within the Wasatch Plateau, which is part of the Colorado Plateau Province. The Wasatch Plateau is a north to south trending highland that overlooks the Castle Valley to the east and the Sanpete Valley to the west. Wild Horse Ridge is a very steep sided, narrow ridge at elevations of 8,700 to 9,050 feet. The project area is drained by the perennial Bear Creek, over a mile west of the project area.

The diversified vegetation consists of grassy sagebrush meadows interspersed with aspen groves and conifer forests, including White and Ponderosa Pine. Some of the understory species included wheat grass, bluegrass, common juniper, shrubby cinquefoil, strawberry, penstemon, mules-ear, needle grass, lupine, manzanita, sagebrush, sedge, currant, and gooseberry.

Previous Research

John Senulis of SENCO-PHENIX performed a file search in the Forest Service Office on July 23, 2003. The following are the previous studies within or near to the project area which meet professional standards:

- 1984, SENCO-PHENIX performed a sample survey of Bear Creek Canyon. No cultural resources were located. (84-1106)
- 1986, Abajo archeology surveyed the non-forest portion of Huntington Canyon south of Rilda Canyon. No cultural resources were located in the vicinity of the current project. (ML 86-9)
- 1990, SENCO-PHENIX surveyed a mine expansion project area in the general project area. No cultural resources were located. (90-263)
- 1991, Sagebrush Archeological Consultants surveyed ten drill holes in the general project area. No cultural resources were located. (91-281)
- 1993, SENCO-PHENIX surveyed 4 drill hole locations in the general project area. No cultural resources were located. (93-549)
- 1994, AERC surveyed potential subsidence zones in Rilda Canyon and elsewhere. No cultural resources were found in Rilda Canyon. (ML 94-734)
- 1997, The Forest Service surveyed restoration areas in Huntington Canyon. No cultural resources were located near the current project area. (ML 97-844)

Methodology

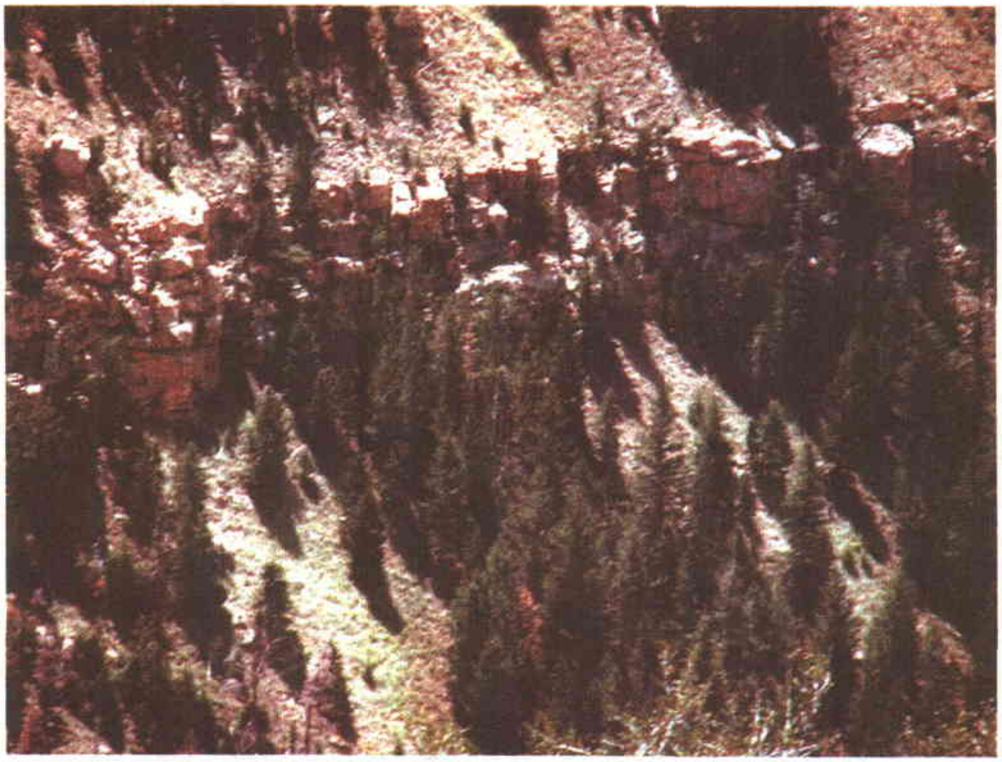
John and Jeanne Senulis of SENCO-PHENIX performed a combination Class III intensive walkover and intuitive survey on August 6, 2003 of the sandstone cliff faces in the potential subsidence area. The policy of the Manti-La Sal National Forest is to survey sandstone cliff faces in areas of potential subsidence, because the cliff faces often collapse when the pillars are removed from the underlying mine. The sandstone cliff faces were examined for the presence of rockshelters, rock art, burials, or other site types that could occur in these outcroppings. Because of the sheer steepness of the outcrop facings, walkover was limited to the areas where foot travel was possible. Some of the rock faces were examined utilizing both binoculars and a camera with a telephoto lens. The east wall of the ridge is a sheer vertical cliff several hundred feet high and could not be surveyed. Portions of the west slopes could be more closely examined, however, some still had slopes in the 70-80 degree range and could not be walked.
All field notes and photographs are on file at the offices of SENCO-PHENIX in Price, Utah.

Findings and Recommendations

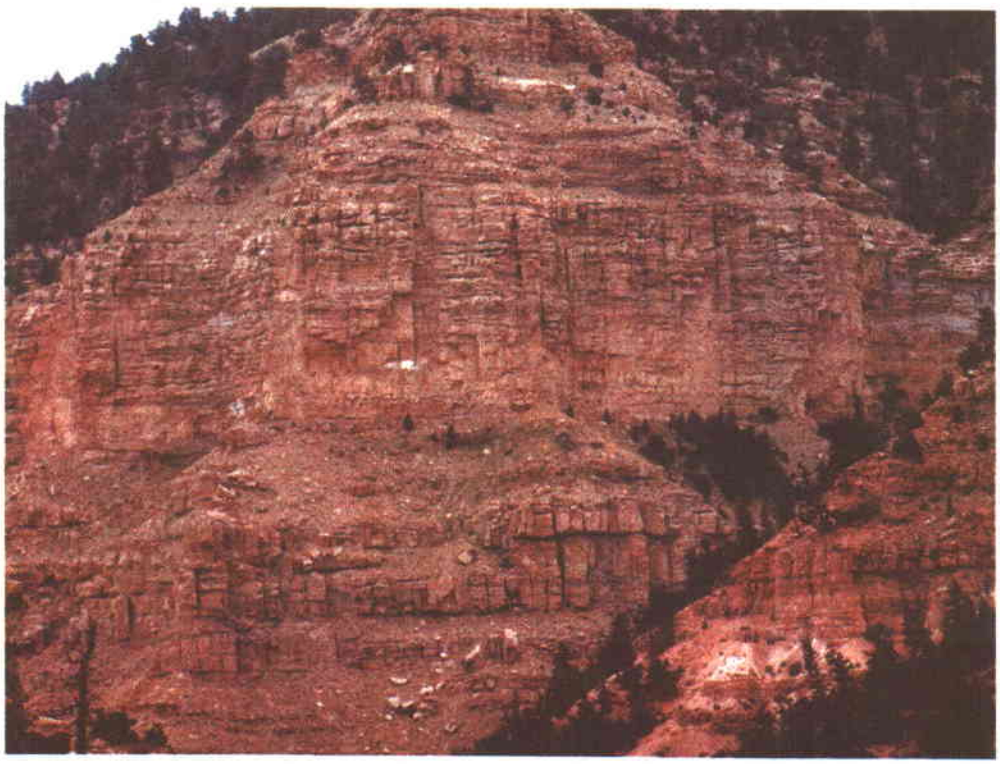
No cultural resources were located and the potential for undetected remains is remote. A finding of no effect is appropriate and archeological clearance without stipulations is recommended.

These recommendations are subject to modification and review by the Manti La Sal Forest Ranger and the Utah SHPO.

Subsidence Project Area



View of the West Face looking Southeast



An Example of the East Wall Steepness

Appendix 4-G

Air Quality Approval Order

State of Utah
DEPARTMENT OF ENVIRONMENTAL QUALITY
DIVISION OF AIR QUALITY

Michael O. Leavitt
Governor

Dianne R. Nielson, Ph.D.
Executive Director

Richard W. Sprott
Director

150 North 1950 West
P.O. Box 144820
Salt Lake City, Utah 84114-4820
(801) 536-4000 Voice
(801) 536-4099 Fax
(801) 536-4414 T.D.D.
Web: www.deq.state.ut.us

DAQE-145-02

February 22, 2002

Charles Reynolds
Co-Op Mining Company
P. O. Box 1245
Huntington, Utah 84528

Dear Mr. Reynolds:

Re: Approval Order: Modification of Approval Order DAQE-862-99, to Increase Production
Emery County - CDS-B, NSPS, Title V, Project Code: N0240-004

The attached document is the Approval Order for the above-referenced project.

Future correspondence on this Approval Order should include the engineer's name as well as the DAQE number as shown on the upper right-hand corner of this letter. Please direct any technical questions you may have on this project to Mr. Enqiang He. He may be reached at (801) 536-4010.

Sincerely,

Richard W. Sprott, Executive Secretary
Utah Air Quality Board

RWS:EH:re

cc: Emery County
Mike Owens, EPA Region VIII

STATE OF UTAH

Department of Environmental Quality

Division of Air Quality

APPROVAL ORDER: MODIFICATION OF APPROVAL ORDER DAQE-862-99, TO INCREASE PRODUCTION

Prepared By: Enqiang He, Engineer
Email: ehe@deq.state.ut.us
(801) 536-4010

APPROVAL NUMBER

DAQE-145-02

Date: February 22, 2002

Co-Op Mining Company
Source Contact
Charles Reynolds
(435) 687-2450

Richard W. Sprott
Executive Secretary
Utah Air Quality Board

Abstract

Co-Op Mining Company has proposed to modify its existing Approval Order (AO) DAQE-862-99, dated October 29, 1999. The source is located in Emery County which is an attainment area for all criteria pollutants. The AO is modified as follows: (1) to increase throughput from 1,100,000 tons/year to 1,950,000 tons/year of coal, and (2) to use an oil burner/space heater in the shop. New Source Performance Standards (NSPS) Subpart Y (Standards of Performance for Coal Preparation Plants) regulations apply to this source. National Emission Standards for Hazardous Air Pollutants (NESHAP) regulations do not apply to this source. Title V regulations apply to this source. The source uses new emission factors for the unpaved haul road. The emissions, in tons per year, will change as follows: PM_{10} (+ 4.88) 13.09, NO_x (+ 0.01) 11.28, SO_2 (+ 0.05) 1.59, CO (+ 0.00) 4.73, and VOC (+ 0.22) 2.88.

The project has been evaluated and found to be consistent with the requirements of the Utah Administrative Code Rule 307 (UAC R307). A 30-day public comment period was held in accordance with UAC R307-401-4 and no comments were received. This air quality Approval Order (AO) authorizes the project with the following conditions, and failure to comply with any of the conditions may constitute a violation of this order.

General Conditions:

1. This Approval Order (AO) applies to the following company:

   Office Location
   Co-Op Mining Company
   P.O. Box 1245
   Huntington, Utah 84528
   Phone Number: (435) 687-2450
   Fax Number: (435) 687-2084

   PLANT LOCATION:

   The equipment listed below in this AO shall be operated at the following location:
   Eleven miles west of Huntington, Utah on state Highway 31

   Universal Transverse Mercator (UTM) Coordinate System: UTM Datum NAD27
   4,361.9 kilometers Northing, 491.7 kilometers Easting, Zone 12

2. All definitions, terms, abbreviations, and references used in this AO conform to those used in the Utah Administrative Code (UAC) Rule 307 (R307), and Title 40 of the Code of Federal Regulations (40 CFR). Unless noted otherwise, references cited in these AO conditions refer to those rules.

3. The limits set forth in this AO shall not be exceeded without prior approval in accordance with R307-401.

4. Modifications to the equipment or processes approved by this AO that could affect the emissions covered by this AO must be approved in accordance with R307-401-1.

5. All records referenced in this AO or in applicable NSPS standards, which are required to

be kept by the owner/operator, shall be made available to the Executive Secretary or Executive Secretary's representative upon request, and the records shall include the two-year period prior to the date of the request. Records shall be kept for the following minimum periods:

A.	Emission inventories	Five years from the due date of each emission statement or until the next inventory is due, whichever is longer.
B.	All other records	Two years.

6. Co-Op Mining Company shall conduct its mining operations in accordance with the terms and conditions of this AO, which was written pursuant to Co-Op Mining Company's Notice of Intent received by the Division of Air Quality (DAQ) on November 13, 2000 and additional information submitted to the DAQ on November 5, 2001.

7. This AO shall replace the AO (DAQE-862-99) dated October 29, 1999.

8. The approved installations shall consist of the following equipment or equivalent*:

Crushers:

South Crusher:
Type: Primary Crushing, Hammer mill
Manufacturer: Unknown
Model: Unknown
Capacity: 200 to 450 tons/hr

Wildhorse Crusher:
Will be same as the South crusher or similar to the Gundlach crusher

Lump Crusher (high ash):
Type: Primary Crushing, Granulator
Manufacturer: Pennsylvania *
Model TK-9-38*
Capacity: 300 tons/hr

Lump Screenage Crusher:
Type: Primary Crushing, 24-inch single Roll Crusher*
Manufacturer: Unknown
Model: Unknown
Capacity: 50 tons/hr

Gundlach Crusher:
Type: Primary Crushing, 4 Roll Crusher
Manufacturer: Gundlach*
Model: 70-DA-1260*
Capacity: 100 to 700 tons/hr

Oversize Crusher:
Type: Secondary Crushing, Square Crusher
Manufacturer: Eagle*
Model: 10210*
Capacity: 50 tons/hr

Screens:

Secondary Picking Screen:
Allis Chalmers, Ripple Flow *
6 x 16 SD
Capacity: 300 to 450 tons/hr

Wildhorse Screen:
Hewitt Robbins *
8x20DD
Capacity: 350 to 900 tons/hr

Upper Screen:
Allis Chalmers, Ripple Flow *
6x16DD
Capacity: 300 to 900 tons/hr

Lower Screen
Allis Chalmers, Low Head *
8x20 DD
Capacity: 350 to 900 tons/hr

Storage Tanks:

One vertical unleaded above ground gasoline storage tank, 11,000 gallon capacity**

One diesel, above ground storage tank, 11,000 gallon capacity**

One diesel, above ground storage tank, 16,000 gallon capacity**

Two above ground Stoker Oil storage tank, 14,000 gallon capacity**

One above ground diesel storage tank**, 1,000 gallon capacity

One above ground diesel storage tank**, 8,000 gallon capacity(Installed in 1999)

One above ground Anti Freeze storage tank, 2,000 gallon capacity

Coal Fired Space Heaters:

Shower House
Stokermatic Warrior* 150/225p, 225,000 BTU/hr
Scale House
Stokermatic Warrior* 150/225p, 225,000 BTU/hr *
Stokermatic Model 2*, 100,000 to 150,000 BTU/hr *
Shop Building
Coal stoker space heater, 250,000 BTU/hr

Miscellaneous Equipment:

Conveyors
Diesel fired Front end loaders
Oil burner/space heater***
Other mobile equipment necessary for facility management

*	Equivalency shall be determined by the Executive Secretary
**	Not subject to 40 CFR Parts 60 Subpart Kb
***	Newly added

Limitations and Tests Procedures

9. Visible emissions from any point or fugitive emission source associated with the installation of the source or with the control facilities shall not exceed 20% opacity. Opacity observations of emissions from stationary sources shall be conducted in accordance with 40 CFR 60, Appendix A, Method 9. Visible emissions determinations for traffic sources shall use procedures similar to Method 9. The normal requirement for observations to be made at 15 second intervals over a six minute period; however, shall not apply. Six points, distributed along the length of the haul road or in the operational area, shall be chosen by the Executive Secretary or the Executive Secretary's representative. An opacity reading shall be made at each point when a vehicle passes the selected points. Opacity readings shall be made ½ vehicle length or greater behind the vehicle and at approximately ½ the height of the vehicle or greater. The accumulated six readings shall be averaged for the compliance value.

10. The production limit of 1,950,000 tons of coal per rolling 12-month period shall not be exceeded.

    To determine compliance with a rolling 12-month total the owner/operator shall calculate a new 12-month total by the twentieth day of each month using data from the previous 12 months. Records of production shall be kept for all periods when the plant is in operation. Production shall be determined by an operations log. The records of production shall be kept on a daily basis.

Roads and Fugitive Dust

11. All unpaved roads and other unpaved operational areas that are used by mobile equipment shall be water sprayed to control fugitive dust. The application of water or chemical treatment shall be used. Treatment shall be of sufficient frequency and quantity to maintain the surface material in a damp/moist condition. The opacity shall not exceed 20% at all times the areas are in use unless it is below freezing. Records of water treatment shall be kept for all periods when the plant is in operation. The records shall include the following items:

    A. Date
    B. Number of treatments made and water quantity
    C. Rainfall received, if any, and approximate amount
    D. Time of day treatments were made

12. Water sprays or chemical dust suppression sprays shall be installed at all drop points at which the material handled may have a moisture content of less than 4% or as needed to comply with the opacity limitations in this AO. Examples of points that require water sprays include the following:

    A. Mine Operations
    B. Storage Piles
    C. Crushing and Screening areas
    D. Conveying Drop Points
    E. Load-out

    The sprays shall operate whenever dry conditions warrant or as determined necessary by the Executive Secretary.

13. The storage piles shall be watered to minimize generation of fugitive dusts, as dry conditions warrant or as determined necessary by the Executive Secretary. Records of water and/or chemical treatment shall be kept for all periods when the plant is in operation.

Fuels

14. The owner/operator shall use only diesel fuel with less than 0.3% sulfur or gasoline in stationary equipment or mobile equipment operated onsite. Sulfur content shall be decided by ASTM Method D2880-71 or D-4294-89, or approved equivalent. The sulfur content shall be tested if directed by the Executive Secretary. The percent by weight of the sulfur contained in the fuel can be obtained from the fuel oil certifications. Certification of fuels shall be either by Co-Op Mining Company's own testing or test reports from the fuel marketer. Records of fuel supplier's test report on sulfur content shall be available on-site for each load delivered. Coal consumption in the Shower house, Scale house and shop building space heaters shall not exceed 65 tons per rolling 12-month period. The sulfur content of the coal shall not exceed 0.60%. Sulfur content in coal shall be decided by ASTM Methods D3177-75 or D4239-85, or approved equivalent.

Federal Limitations and Requirements

15. In addition to the requirements of this AO, all provisions of 40 CFR 60, NSPS[1] Subparts A and Y, (Standards of Performance for Coal Preparation Plants), apply to this crushing facility. The storage tanks are not subject to 40 CFR 60 subpart Kb.

Records & Miscellaneous

16. At all times, including periods of startup, shutdown, and malfunction, owners and operators shall, to the extent practicable, maintain and operate any equipment approved under this Approval Order including associated air pollution control equipment in a manner consistent with good air pollution control practice for minimizing emissions. Determination of whether acceptable operating and maintenance procedures are being used will be based on information available to the Executive Secretary which may include, but is not limited to, monitoring results, opacity observations, review of operating and maintenance procedures, and inspection of the source. All maintenance performed on equipment authorized by this AO shall be recorded.

17. The owner/operator shall comply with R307-150 Series. Inventories, Testing and Monitoring.

18. The owner/operator shall comply with R307-107. General Requirements: Unavoidable Breakdowns.

The Executive Secretary shall be notified in writing if the company is sold or changes its name.

This AO in no way releases the owner or operator from any liability for compliance with all other applicable federal, state, and local regulations including R307.

A copy of the rules, regulations and/or attachments addressed in this AO may be obtained by contacting the Division of Air Quality. The Utah Administrative Code R307 rules used by DAQ, the Notice of Intent (NOI) guide, and other air quality documents and forms may also be obtained on the Internet at the following web site: http://www.eq.state.ut.us/eqair/aq_home.htm

The annual emission estimations below include space heaters, conveyors and loadout, crushing and screening, haul road, storage piles, and diesel storage tank emissions. These emissions are for the purpose of determining the applicability of Prevention of Significant Deterioration, nonattainment area, maintenance area, and Title V source requirements of the R307. <u>They are not to be used for determining compliance</u>.

The Potential To Emit (PTE) emissions for this source (the Bear Canyon Mine) are currently calculated at the following values:

[1] NSPS = New Source Performance Standards.

	Pollutant	Tons/yr
A.	PM_{10}	13.09
B.	SO_2	1.59
C.	NO_x	11.28
D.	CO	4.73
E.	VOC	2.88

Approved By:

Richard W. Sprott, Executive Secretary
Utah Air Quality Board

Chapter 5

R645-301-500 Engineering

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R645-301-500 Engineering

R645-301-511 General Requirements

This chapter includes descriptions of existing and proposed coal mining and reclamation operations and their potential impacts to the environment.

R645-301-512 Certification

Maps, cross-sections and designs required by R645-301 to be certified have been prepared by or under the direction of a qualified, registered, professional engineer whose stamp and signature can be found on the individual maps or design. All maps, cross-sections, and designs will meet all requirements of R645-301.

R645-301-513 Compliance with MSHA Regulation and Approvals

C. W. Mining Co. agrees to comply with all MSHA regulations pertaining to dams, embankments, impoundments, sediment ponds, excess spoil, coal waste, portals, and mining operation, timing, and sequence.

R645-301-514 Inspections

All engineering inspections will be conducted in accordance with both Utah State Law and the Federal Code of Regulations. The inspections will be done by either a qualified registered engineer or a qualified person under the direction of a qualified registered engineer as required by law.

R645-301-515 Reporting and Emergency Procedures

515.100 Slides

At any time a slide occurs which may have a potential adverse effect on public property, health, safety, or the environment, the Division will be notified by the fastest available means and C.W. Mining Co. will comply with any reasonable remedial measures required by the Division.

515.200 Impoundment Hazards

If any examination or inspection discloses that a potential impoundment hazard exists, C. W. Mining Co. will promptly inform the Division of the finding and of the emergency procedures formulated for public protection and remedial action. If adequate procedures cannot be formulated or implemented, the Division will be notified immediately.

515.310 Temporary Abandonment

In the event of a temporary abandonment of operations C. W. Mining will still comply with all provisions of the approved permit.

515.311 Inactive Areas

C. W. Mining will support and maintain all access openings and surface facilities in all areas where there are no current operations, but operations are to be resumed.

515.320 Temporary Cessation of Operations

In the event of a temporary cessation of operation for a period of 30 days or more, C.W. Mining will submit a notice to the Division as soon as it is known that the temporary cessation will extend beyond 30 days or more. This notice will include: a statement of the exact number of surface acres and the horizontal and vertical extent of sub-surface strata which have been in the permit area prior to cessation or abandonment, the extent and kind of surface reclamation of surface area which will have been accomplished, and identification of the backfilling, regrading, revegetation, environmental monitoring, underground opening closures and water treatment activities that will continue during temporary cessation.

C.W. Mining will seal portals which are not to be utilized for mine inspection or access during temporary cessation of operation. These seals will be constructed of woven wire or cement block and securely attached to the portal entry so as to make trespass by men or animals prohibitive. All portals which are to be utilized will be posted with "No Trespassing" and "Keep

Out" notices. Where doors exist such as fan entries, this will also be locked and signed accordingly.

R645-301-520 Operation Plan

C. W. Mining Company has operations at several different sites. Because of the complexity of the entire unit, 5 different areas of operation and reclamation will be discussed. The five areas are listed below.

C. W. Mining Company Permit Area Components

1. Bear Canyon Mine Area
2. Tank Seam Area
3. WHR Blind Canyon Seam Area
4. WHR Tank Seam Area
5. Morhland Area

521.100 Cross Section and Maps

Maps showing all relevant information have been prepared by or under the direction of, and are certified by a registered certified professional engineer.

521.110 Previously Mine Area

The location and extent of know workings of active, inactive, and abandoned underground mines, are shown on Plates 5-1.

521.120 Surface and Subsurface Facilities and Features

The location of all buildings, surface and sub-surface man made features, public roads, waste piles, sediment ponds, and water impoundments are shown on Plates 5-2.

521.130 Landownership and Right of Entry and Public Interest Maps

521.131 Land Ownership Maps

All boundaries of lands and names of present owners of record of those land, both surface and subsurface are shown on Plate1-2 and Plate 1-3.

521.132 Boundaries of Land C.W. Mining Has Right of Entry To.

These boundaries are shown on Plate 1-2 and Plate 1-3.

521.133 Protection of Public Interest

C. W. Mining currently has no operations within 100 feet of public land or a public road.

521.140 Mine Maps and Permit Area Maps

521.141 Disturbed Boundary and Timing Of Mining

The boundary of all disturbed and proposed disturbed areas are shown on Plates 5-2. The areas to be mined and the sequence and timing of the mining are shown on Plate 5-1A (Blind Canyon Seam), Plate 5-1B (Hiawatha Seam), and Plate 5-1C (Tank Seam). Plates 5-2 also shown the changes to all facilities and features that have been or will be made. The boundary showing all areas that may be affected by mining is shown on Plate ~~2~~1-1 (Permit Boundary).

521.142 Underground Workings and Subsidence Areas

Underground workings and areas where planned subsidence mining methods will be used are shown on Plates 5-1. R645-301-525 discusses subsidence in greater detail.

521.143 Mine Waste Disposal Sites

Temporary and permanent waste disposal sites will be shown on Plates 5-2. C. W. Mining Company is currently permitted to haul all mine waste to Hiawatha for permanent disposal. The location of these sites can be found in the Hiawatha Coal Company MRP.

521.150 Land Configuration Maps

All land surface configuration maps give surface contours which adequately represent the existing land surface configuration of the permit area.

521.151 Cross Sections and Slope Measurements

All areas have representative cross sections that extend 100 feet beyond the disturbed boundary and show the slope measurements for pre-mining, operational, and post-mining land configurations. The cross-sections are discussed in greater detail in R645-301-521.160 and R645-301-540.

521.152 Previously Mined Areas

See R645-301-521.151

521.160 Maps and Cross Sections of the Features of the Permit Area

Plates 5-2 show the location of all buildings, utility corridors, facilities, affected areas, coal storage areas, cleaning and loading areas, topsoil stockpile areas, coal preparation and storage areas, underground development waste areas, non coal waste areas, waste disposal facilities, refuse piles, slurry impoundments, sediment ponds, and explosive storage magazines. Cross section details are shown on Plate 5-8 and in Appendices 5-H, 5-I, 5-J, 5-K, and 5-L The location of the cross sections are shown on Plates 2-3, 5-6, and 5-7.

521.170 Transportation Facilities Maps

Transportation facilities are shown on Plates 5-2. Road details are shown on Plates 5-4 and on the cross-sections in Appendices 5-H, 5-I, 5-J, 5-K, and 5-L. Road construction details can be found in Appendices 5-F and 5-G.

521.180 Support Facilities

Support Facilities are discussed under R645-301-526.

521.200 Signs and Marker Specifications

Signs and markers will be posted, maintained and removed by C. W. Mining Company and will be of a durable material and be a uniform design that can be easily seen and read. The will be maintained during all activities to which they pertain and will conform with all local laws and regulations.

521.240 Mine and Permit Identification Signs

Identification signs are displayed at each point of access from public roads to areas of surface operations and facilities on the permit area. These signs show the name, business address and telephone number of C. W. Mining Company and the identification number of the permanent program permit authorizing coal mining and reclamation activities.

521.250 Perimeter Markers

The perimeter of all areas affected by surface operations or facilities are clearly marked by perimeter markers.

521.260 Buffer Zone Markers

Stream buffer zone signs are installed at locations where mining and reclamation operations are conducted in the vicinity or perennial and intermittent streams.

521.270 Topsoil Markers

Topsoil markers have been installed where topsoil or other vegetation-supporting material is physically segregated and stockpiled under R645-301-234.

R645-301-522 Coal Recovery

It is in the interest of C. W. Mining Company to maximize the recovery of coal resources. A large portion of C. W. Mining's coal resources are contained in Federal leases. A major condition of each lease agreement is maximum recovery of resources. When accessible, mine workings in each lease are inspected on a regular basis by the Bureau of Land Management personnel experienced in underground coal mining methods. Justification for not recovering coal deposits that may be detrimentally affected in terms of future recovery by the proposed operations include the following.

A. Seams that are too thin to be economically minable given existing or reasonably foreseeable technology.

B. Coal seems separated by insufficient rock intervals to allow safe mining above or below worked out areas.

C. Seems that are relatively thick but not extensive, and isolated by thin coal which would make development cost prohibitive.

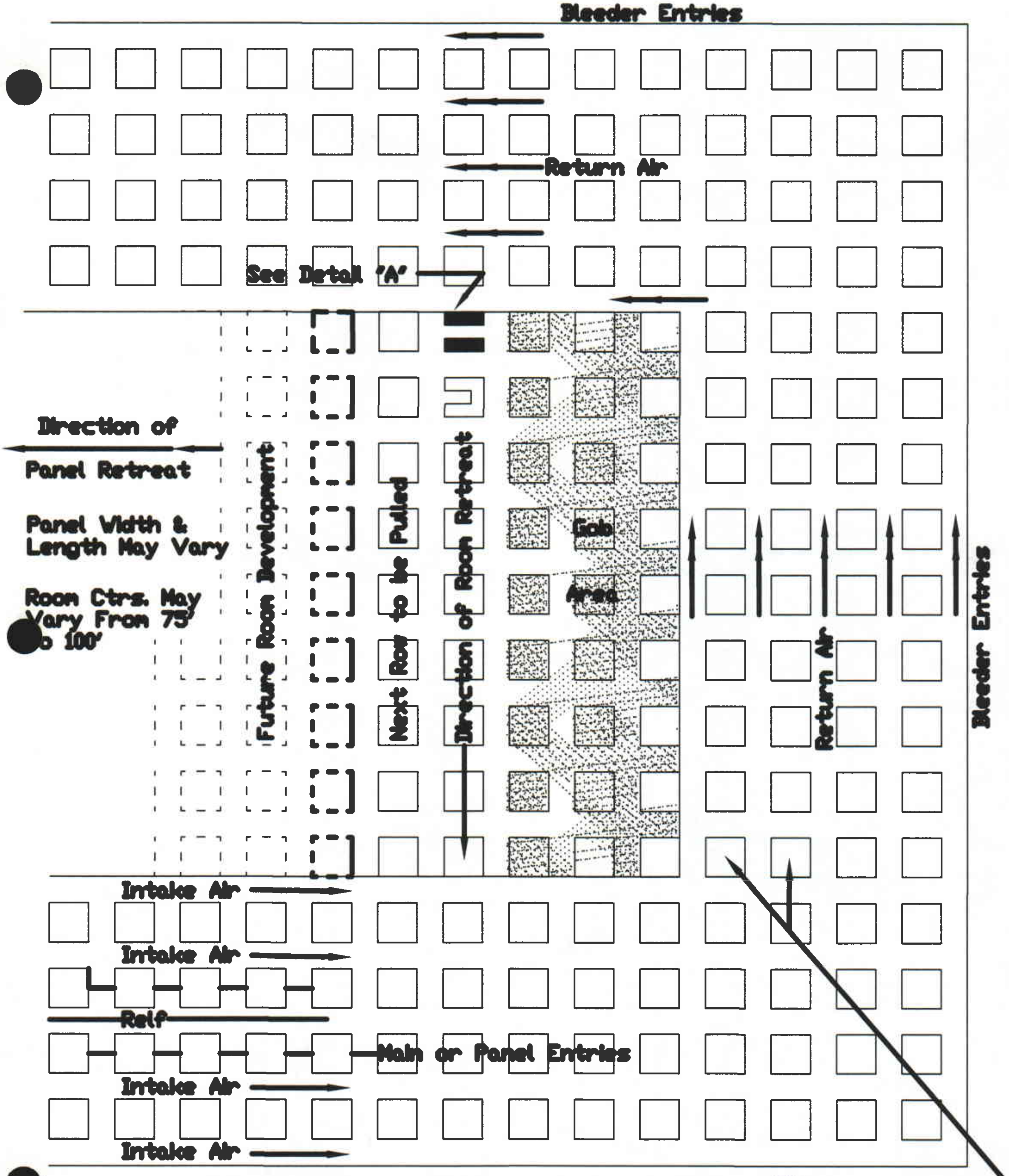
There are four main seams in the Bear Canyon Property, the Tank Seam, the Bear Canyon Seam, Blind Canyon Seam, and Hiawatha Seam. ~~There are no plans to mine the upper Bear Canyon seam~~ There are no plans to mine both seams due to the close proximity of the Bear Canyon seam to the Blind Canyon Seam (<30 feet interburden). There are no plans to mine the Hiawatha Seam in Wild Horse Ridge due to the thinning of the seam. Mining plan, sequence and projected development for the Bear Canyon, Hiawatha, and Tank seams are shown on Plates 5-1A, 5-1B and 5-1C respectively. Geology information is discussed in Chapter 6.

R645-301-523 Mining Method

Mining at the Bear Canyon complex is done by a longwall and continuous miners. ~~The miners discharge into shuttle cars, which carry the coal to the feeder breaker. The feeder breaker discharges the coal onto the belt conveyor where it is taken out of the mine.~~ The main entries consist of a five-entry system on 80 ft -200 ft centers to be driven to the property limits. For longwall recovery 2-5 gate entries are driven off the mains on either side of the panel to the head of the panel where they are connected by bleeders. The longwall then mines out the panel. For continuous miner recover sub-mains consisting of five entries on 80 ft - 200 ft centers are then driven off the mains and room-and-pillar panels are developed off the sub-mains. Rooms are developed within the panels on 70 ft - 150 ft centers. This is referred to as "Development". The pillars are then recovered according to the approved plan. This is referred to as "retreat". Timber or mobile roof supports are installed to support the roof and provide for breaker control of the caving roof. Retreat mining of this type will provide a recovery of 70pct - 80 pct within the panels. See Figures 5-1 and 5-2. Sub-mains under the escarpment area in Bear Canyon will be developed and left.

Anticipated average annual production is 2,100,000 Tons from the longwall face and 400,000 Tons from development mining. Before the longwall face comes on line and after it is finished some room and pillar retreat mining will be done. The average annual production from room and pillar retreat mining is 600,000 Tons.

As can be seen on Plates 5-1A and 5-1B, the lower seam workings are planned to be columnized with the upper seams as closely as practicable. Where this is not practiced due to geologic conditions, pillars will be adequately sized to afford stability for the rooms. Geologic conditions and the limited lateral extent of reserves in the Tank Seam precludes columnizing of pillars with the other seams in some areas. However, experience has shown that the overburden (250') between these seams will provide adequate roof stability even if the pillars are not all columnized. The mining plan sequence allows for recovery of the upper seam areas (Tank Seam first, then Blind Canyon Seam) prior to final recovery of the lower seam. This procedure is consistent with accepted engineering practice in multiple seam mining.

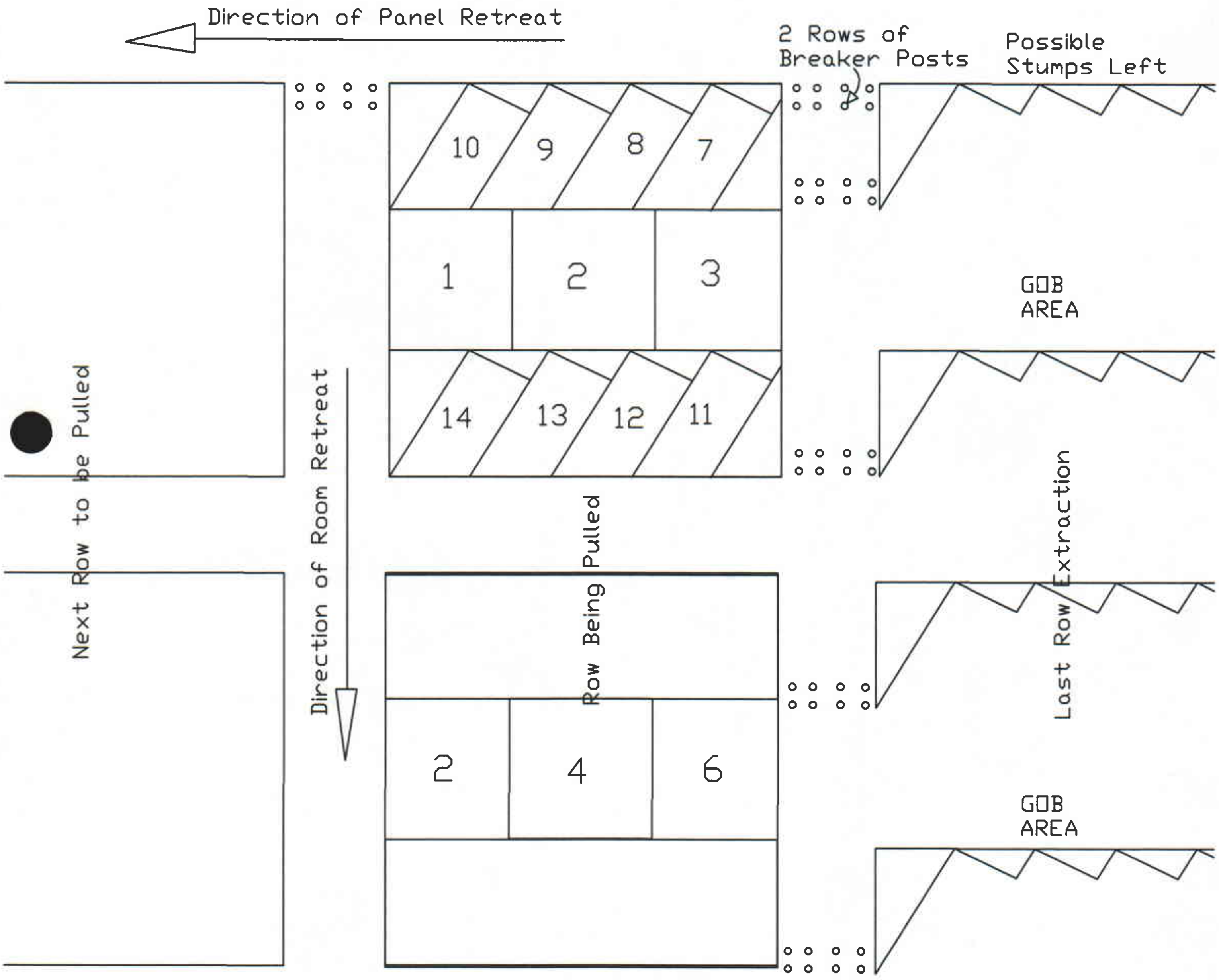


Note: 2 Rows of Bleeder Entries may be pulled if conditions allow.

Scale: 1 in. = 200 ft

Figure 5-1 Typical Panel Recovery

Note: Room centers may vary from 75' to 100'.
Typical cut sequence is as shown. May vary due to condition



Scale: 1 in. = 200 Ft.

Figure 5-2 Typical Pillar Extraction

Ventilation, Water System, Dust Suppression, and Dewatering

Ventilation. A separate fan with adequate capacity supplies air to each of the mines. Intake air is carried in the entries on one side of a set, while return air is carried through the entries on the other side. Air is directed through the mine by stoppings, doors, overcasts, regulators and brattice cloth. After sweeping the working faces of each section the air is directed into the return air courses and out of the mine. Little or no methane gas is generated in any of the Blackhawk seams.

Water System. Water generated in the mine is collected in sumps and used at the mine. Some water from the roof is collected and used as culinary water outside for use in the bathhouses and as drinking water in the offices (Appendix 5-B). Tests for portability are required and made quarterly.

Dust Suppression. Water also serves for sprays on the machines at the working faces, on the coal at belt heads, transfer points and for the tipple for dust suppression.

Dewatering. Water generated is used or contained within the mine, and excess water is discharged to Bear Creek. Mine dewatering is governed by the mine discharge permit (Appendix 7-B).

Equipment Selection

Co-Op will utilize the equipment described in the following list for its mining operation and will acquire any additional equipment as required to maintain a sound mining operation.

Surface Equipment
- vibrating screens
- crushers
- conveyors
- front end loaders
- road grader
- crawler tractor
- fork lift

Recovery Rate

The recoverable coal reserves were estimated by multiplying the in-place reserve by a recovery factor of 70 pct.

The operation will produce 750,000 to 2,000,000 tons of raw coal per year with 2 to 4 miner sections working ~~240~~ 360 days. This is 3,000+ to 8,000+ tons per day with 2 to 4 production shifts operating. The rate of production (considering a lower rate during the initial buildup years plus the tonnage still to be mined in the area of old workings) will make the projected mine life approx 29 years.

Table 5-1 Coal Reserves - Bear Canyon Mine

Reserve Area	Seam	Coal Reserves (tons)	
		In-Place	Recoverable
Federal Lease			
U-46484	Blind Canyon	5,093,206	1,586,852
	Hiawatha	7,015,758	3,085,990
	SUBTOTAL	12,108,964	4,672,842
(U-61048)	Tank	490,470	0
	Hiawatha	7,540,784	2,942,416
	SUBTOTAL	8,031,254	2,942,416
(U-61049)	Tank	10,027,191	6,433,459
	Blind Canyon	0	0
	Hiawatha	18,428,989	11,822,475
	SUBTOTAL	28,456,180	18,255,934
(U-024318)	Blind Canyon*	~~65,363~~ 52,763	~~12,600~~0
	Hiawatha	0	0
	SUBTOTAL	~~65,363~~52,763	~~12,600~~0
(U-024316)	Tank	~~694,500~~1,099,006	~~347,250~~ 0
	Blind Canyon	~~Unknown~~1,112,202	~~Unknown~~366,089
	Hiawatha	~~Unknown~~1,770,113	~~Unknown~~959,124
	SUBTOTAL	~~694,500~~3,981,321	~~347,250~~1,325,213
(U-020668)	Tank	~~448,312~~600,036	~~224,156~~318,706
	Blind Canyon	~~3,185,699~~2,647,894	~~1,592,850~~774,205
	SUBTOTAL	~~3,634,011~~3,247,930	~~1,817,005~~1,092,911
(U-38727)	Tank	~~4,555,325~~4,300,059	~~2,227,662~~2,958,627
	Blind Canyon	~~3,558,827~~3,779,496	~~1,592,850~~1,468,042
	SUBTOTAL	~~8,114,152~~8,079,555	~~4,057,075~~4,426,669
Fee Land	Tank	~~7,304,476~~7,836,811	~~3,652,238~~3,986,939
	Blind Canyon	~~9,212,699~~9,395,909	~~4,606,350~~3,808,970
	Hiawatha	2,568,465	1,284,232
	SUBTOTAL	~~19,085,640~~19,801,185	~~9,542,820~~9,080,141
Total ~~Area~~ Tons	Tank	~~13,002,613~~24,353,573	~~6,501,306~~13,697,731
	Blind Canyon	~~16,022,588~~22,081,470	~~7,991,213~~8,004,158
	Hiawatha	~~2,568,465~~37,324,109	~~1,284,232~~20,094,237
TOTAL		~~31,593,666~~83,759,152	~~15,776,751~~41,796,126

NOTES: 1. Reserves based on latest projections (7/21/06~~2/28/97~~) submitted to the B.L.M. in the ~~L.M.U. Application~~ R2P2 update.

2. Current permit application will allow for mining of Lease U-024316 in the Tank Seam only until additional hydrologic and geologic information can be obtained.

- Blind Canyon Seam reserves mined out in 1984-1985.

R645-301-524 Blasting and Explosives

C. W. Mining Company commits to follow all of the regulations in Section R645-301-524 that apply to its operation.

R645-301-525 Subsidence Control Plan

Subsidence monitoring points are shown on Plate 5-3. Appendix 5-C contains the subsidence monitoring and control plan.

525-300 Subsidence Control

Subsidence control is outlined below and in Appendix 5-C

Barrier Pillars

Pillars of coal generally are left underground to protect surface or underground features which must be maintained and protected for the life of the mine (main entries) or permanently (oil or gas wells). The size of some is specified by law; others are designed by the operator to provide the protection needed. ~~Submains under the escarpment area in Bear Canyon will be left.~~

Property Boundaries

Area boundaries of individual leases and fee property are shown on Plate 1-3. Protection of boundaries and property adjacent to the permit area is provided by continuous barrier pillars a min of 100 ft wide. This is wide enough to prevent subsidence across the boundary resulting from angle of draw. On the north end of the permit boundary no barrier will be left because this area is adjacent to the Hiawatha permit boundary which is part of the Bear Canyon LMU so mining could potentially cross from one permit boundary into the other uninterrupted.

Outcrop Protection

In most areas, coal outcrops are buried and are not visible from the surface. Outcrops are either covered to some depth with overburden or, in many areas; the coal has been burned for some distance from the surface. Where neither of these situations exist, routine tests of the coal may show that it has been "weathered" or "oxidized" and mining will be stopped within 200 ft of the outcrop.

Barrier pillars to protect main and sub- main entries have been made large enough (100 ft or greater) to assure protection of the entries for their useful life. When the area serviced is mined out, entry pillars will be recovered on the way out.

Protection of Natural Surface Structures & Streams

~~C. W. Mining's commitment to maintain a min of 200 ft barrier pillars to outcrops where required by lease stipulations, or protection of streams and wildlife to minimize the possibility of escarpment failure and resulting detrimental impacts to down stream water quality or nesting raptor. Submains under the escarpment area in Bear Canyon will be left unless otherwise approved, no retreat mining will take place under the escarpment areas, which are outside of the potential subsidence zones shown on Plate 5-3.~~ The primary natural structures that need protection are escarpments and streams. Escarpment locations are shown on Plate 5-3 and 5-3A, and a discussion of their protection is included in Appendix 5C. The only stream channels which lies over the minable portion of the permit area is Bear Creek, where it flows through Federal Lease U-024316, and Fish Creek where it flows through a portion of Federal Lease U-61049 and

In areas where coal burn exists the burning of the coal as caused natural subsidence causing failure of some natural structures. A barrier left adjacent to these areas would cause an interruption between the natural and man made subsidence creating greater impacts to the surface. Because of this no barrier will be left in these areas unless it is needed for roof stability or temperature considerations, in which cause the minimum possible size will be used.

Protection of Manmade Features (Surface & Subsurface)

Man made features and structures which exist on the minable portion of the permit area consist of a hunting lodge which exists in the Wild Horse Ridge Area. There are some forest trails but they are all located beyond the coal outcrops. Maximum coal recovery in the controlled uniform manner planned for this mine should result in even surface subsidence with minimal disturbance.

Buildings within 1,000 ft of Permit Area. A hunting lodge lies within 1,000 ft. of the permit area. Adequate barrier protection will be provided to prevent subsidence of this structure. No buildings have been identified above the potential subsidence zone.

Existing Public Roads. The main access road to the property is a public road. It provides access from Huntington Canyon to the mine. The lease agreement between C.W. Mining and C.O.P Coal Development requires C.W. Mining to allow access through the mine site for representatives of the property owners and Forest Service. The access road to the Wild Horse Ridge area consists of an existing road used to access a private hunting cabin which is owned by Sportsman's, the lessee of the hunting rights to the property. The main road is posted with speed control and general traffic control signs. When mining has been completed, the roads which are not approved to remain for post-mining land use will be reclaimed.

Protection of Oil, Gas and Water Wells

There are no active or abandoned oil or gas wells within the permit boundary.

R645-301-526 Mine Facilities

526.100 Existing Mine Structures and Facilities

This is addressed in Appendix 5-A

526.200 Utility Installation and Support Facilities

The only public utilities located on or near the mine site are Big Bear Spring and Birch Spring. C. W. Mining will monitor these springs for any adverse affects related to the mining operation as described in Chapter 7 R645-301-731.200

526.300 Water Pollution Control Facilities

This is addressed in R645-301-732, R654-301-733, R654-301-738, and R645-301-744.

526.400 Air Pollution Control Facilities

This is addressed in R645-301-420.

R645-301-527 Transportation Facilities

There are eleven primary roads in the permit area: Bear Canyon Haul Road, Portal Access Road, Tank Seam Access Road, Shower House Road, road to Sediment Pond A, Tipple Access Road, Shop Road, No. 3 Mine Access Road, No. 4 Mine Access Road, and the No.1 and No.2 Conveyor Access Roads. All roads are shown on Plates 5-2. Road profiles and typical cross sections are shown on Plates 5-4. A description of all roads is included in Appendix 5-F, along with maintenance procedures. Construction of the Tank Seam Access Road is discussed in Appendix 5-G. Construction of the Wild Horse Ridge road is discussed in Appendix 5-J. Construction of the Wild Horse RidgeTank Seam Road is discussed in Appendix 5-K. Construction of the Mohrland Road is discussed in Appendix 5-L. Construction of the No. 4 Mine Access Road is discussed in Appendix 5-K.

The mine area is approached on the Bear Canyon Haul Road. The #1 mine portal is reached on the Portal Access Road. The Tank Seam Portal is reached on the Tank Seam Access Road. The Wild Horse Ridge area is accessed on the No. 3 Mine Access Road. The WHR Tank Seam is accessed on the No. 4 Mine Access Road. Six other primary roads provide access; to the Sediment Pond A, the coal preparation facility (tipple), the shop, the Wild Horse Ridge conveyor belts, and to the Shower House.

The Bear Canyon Haul Road, No. 3 Mine Access Road, and part of the No. 4 Mine Access Road are also used by customers of Sportsman's Hunting to access a hunting cabin, which exists in the right fork of Bear Canyon. This non-mining recreational use of the road occurs primarily from May until November, typically 2-3 times per week. A lease agreement between Sportsman's and C.O.P.D. provides them with a right-of-way access to the cabin in Bear Canyon.

There is one ancillary (jeep trail) road shown near the portal on Plate 5-2C, which was in existence prior to C. W. Mining's activities and has not been used by C. W. Mining for mining and reclamation activities.

Traffic on the haul road consists primarily of coal haul trucks, mine personnel passenger vehicles, vendor cars and trucks, and other passenger and recreational vehicles. The traffic on the shop and portal access roads and No. 3 Mine access road consists of supply tractors, vendor trucks and company vehicles, with some additional passenger and recreational vehicular use.

Roads are maintained in such a manner that the performance standards will be met throughout the life of the entire transportation facility, including maintenance of the surface, shoulders, parking and side areas, and erosion control structures for safe and efficient utilization of the road. In the event a road is damaged by a catastrophic event, such as a flood or earthquake, the road will be repaired as soon as practical after the damage has occurred.

The road surfacing material for the No. 3 Mine Portal Access Road, No. 4 Mine Access Road, and Wild Horse Ridge Conveyor Access Roads will consist of in-place material and/or road base (gravel) material. This material has proven adequate on the other Bear Canyon Mine Access Roads for the type of traffic which will use the roads, consisting of pickup trucks and mine vehicles and tractors. Soil analyses of the in-place material for the Wild Horse Ridge area is presented in Appendix 2-F and Appendix 2-G. Soil analysis of the road base material was analyzed during the analysis of in-place plant growth material, presented in Appendix 2-D. This material is represented by the soil samples identified as SHP-1, SHP-2 and PAR-1 (Soil segment from 0-0.5 ft.). Sample locations are shown on Plates 2-3B, 2-3C and 2-3D. None of the investigations revealed any acid- or toxic- forming materials. Any coal waste material encountered during the portal face-up process will be treated as described in R645-301-528, and will not be used in the fill or as road surface material.

Reclamation of roads and parking areas is treated in the same manner as other working areas. Any asphalt or treated surfaces will be removed prior to rehabilitation upon completion of mining. Roads which are permitted to remain in place will be fit with drainage control structures adequate for post-mining use. See Plates 5-4, 5-5 and 5-6, and road agreements under Appendix 5-F.

R645-301-528 Handling and Disposal of Coal, Overburden, Excess Spoil, and Coal Mine Waste

528.100 Coal Removal, Handling, Storage, Cleaning and Transportation Areas and Structures

Coal is carried from the mines by conveyor belt to surge bins, and then conveyed to the sizing and crushing plant (tipple), where the lump is removed and diverted to the lump bin or seasonal storage area. The rest of the oversized is crushed, and then sized to meet the various requirements of different customers. It is then conveyed to the truck load-out bins or the stockpile area.

Coal will be transferred from the Tank Seam to the Blind Canyon Seam of the Bear Canyon No. 1 Mine through a 7 ft diameter borehole, which has been bored from the surface adjacent to the portal. The conveyor from the portal and the drop tube structure will be enclosed. Air cannons will be placed on the outside of the drop tube to prevent the wedging of coal in the tube. A 7 ft diameter borehole will be used in Wild Horse Ridge to transfer coal from the Tank Seam (#4 Mine) to the Blind Canyon Seam (#3 Mine). This borehole, however, will be constructed underground.

Coal from the #3 mine will be transferred from the portal area to the tipple using an overland conveyor. This conveyor is shown on Plates 5-2C, 5-2F and 5-2G. The conveyor will either be suspended in the air from cables or set on stands on the ground. Because the Wild Horse Ridge area is an important migratory path for the deer and elk, restriction of this migration by the conveyor in areas where it sets on the ground is a concern. To mitigate this potential problem, the stands will be constructed to suspend the bottom of the conveyor belt a minimum of 36" above the ground. A typical cross-section of the conveyor is shown in Appendix 7-K, Figure 1. All moving parts are contained within the spill pan, so there will be no danger to wildlife passing under the conveyor.

528.200 Overburden

Overburden was removed from areas in the vicinity of mine portals and used in the construction of the portal pads. The material will be utilized to reclamation to cover the highwalls and to reshape the slopes to approximate their pre-mining condition as described under R645-301-540.

528.301 Excess Spoil

No excess spoil is proposed to be generated during the term of this permit. C. W. Mining commits to not conduct any activities that could generate excess spoil unless a plan is submitted and approval granted by the Division.

528.320 Coal Mine Waste

Coal mine waste such as separated waste rock will be temporarily stored at the designated site on the main storage pad shown on Plate 5-2C. Storage time in this area will not exceed 15 days. Each time waste rock is placed in the area, a survey will be conducted to determine the approximate volume of material being placed in the designated area. A log will be kept of the survey which includes a sketch of the material pile and placement dates and will be available onsite for review. In-Mine roof sample analyses are included in Appendix 6-C. A maximum volume of 150 yd^3 of material will be stored in the temporary area at any given time.

The material will then be returned underground and either crushed prior to transportation to use as underground road base material, or placed underground in dry areas in accordance with MSHA regulations. Samples will be taken during future development in accordance with R645-301-623.100.

Roof and rock materials developed underground during mining and related tunneling activities will be placed underground. When relocation of these materials is required underground, they will be placed in "dry" areas where there are no active seeps, sumps or drippers.

Bear Canyon #3 and #4 Mines. Coal Mine Waste such as separated waste rock which is generated from the Wild Horse Ridge operation and can not be used as described above will be hauled to Hiawatha (C1/007/011) and placed in ~~Slurry Pond 5A~~ Refuse pile 1 (MSHA ID# 1211-UT-09-~~00098~~ 02157-04). All material placement will comply with the requirements of the Hiawatha Coal Company's Coal Mining and Reclamation Plan, Section R645-301-528.

Prior to being shipped to the Hiawatha Mine, waste material will be tested for acid and toxic properties in accordance with Table 5O-1. Any material found having acid and toxic properties will be disposed of in the Hiawatha ~~Slurry Pond 5A~~ Refuse pile 1 in accordance with the Hiawatha MRP requirements for acid- and/or toxic-forming material. For sediment control, runoff from any material placed in the temporary storage area reports to and is contained in sediment pond "A".

528.321 Return of Coal Processing Waste to Abandoned Underground Mines

No coal processing waste has been, or is proposed to be, disposed of in abandoned underground mine workings. No coal processing waste will be disposed of in abandoned underground mine workings without the express approval of the Division and MSHA.

528.322 Refuse Piles

C. W. Mining Currently has no permanent refuse piles.

528.323 Burning and Burned Coal Mine Waste

MSHA has verbally notified Co-Op Mining Company that a fire extinguishing plan for the temporary rock pile is not required by their office, since the area utilizes minimal storage, which is temporary. In the event a fire does occur, however, it will be extinguished by spreading the material out on the storage pad and allowing it to burn out and/or distinguishing the fire using water from the belt tower water supply lines.

528.330 Non-Coal Mine Waste

Non-coal waste generated in the operation of the mine is placed in metal dumpsters which are strategically located on the property. A local trash collector is contracted to replace these bins when they are near capacity. Appendix 5-D addresses a comprehensive plan to handle toxic or contaminated material.

528.340 Underground Development Waste

Historically there has been a minimal amount of underground development waste produced. This waste has been associated with the development of portal entries or vent shafts and in each case the waste has been used in the construction of the portal sites or used within the mine to fill low areas. This same process will be followed for all future development waste.

528.400 Dams, Embankments and Other Impoundments

C. W. Mining currently has only 4 dams, embankments and impoundments. All 4 of them are sediment ponds. Three sedimentation ponds are used to control the main Bear Canyon area (Plate 7-1B), a fourth sedimentation pond on the Bear Canyon #3 mine portal pad, and additional ASCA structures. A slope stability analysis for Sediment Pond "A" is shown in Appendix 7-E. The properties of the embankment material for the three ponds are similar due to the close proximity of the three ponds to each other and the native material being used for the pond embankments. The slope stability analysis, which would apply to ponds "B" and "C" also, shows that the minimum safety factor for the pond slopes is 2.8. This is calculated using a slope of 2H:1V. Slopes will be constructed at a slope of 2H:1V or less, providing a 2.8 safety factor. This is greater than the minimum required safety factor of 1.3. These facilities will be maintained as long as it is required to meet the effluent limitations of applicable federal or state laws for runoff or drainage. When their usefulness is ended, they will be removed and the sites reclaimed as described previously.

The fourth pond treats the #3 mine portal pad. A slope stability analysis for the embankment material in this area is shown in Appendix 5-J, pg. 3O-32. This embankment will also be constructed at a slope of 2H:1V, providing a minimum safety factory of 1.46, which is greater than the required safety factor of 1.43.

In areas where sedimentations ponds are not used, additional ASCA controls are used. These controls are discussed in detail in Appendix 7-K.

R645-301-529 Management of Mine Openings

Exploratory Holes, Bore Holes, And Wells. Upon abandonment of drilling operations, all drill holes are to be cemented with an approved slurry. The slurry mixture will consist of 5.2-5.5 gal. of water per bag of cement. C.W. Mining is committed to plugging all openeings as described in R645-301-551

Shafts. The shafts will be filled from bottom to collar with non-combustible material. A cap consisting of a 6 inch thick reinforced concrete slab will be used as a seal. The cap will be equipped with a 2 in. dia vent pipe and will extend for a distance of 5 ft below the surface of the shaft collar.

Mine Entries. Seals will be installed in all entries as soon as mining is completed and the mine is to be abandoned. Seals will be located at least 25 ft inside the portal mouth entry. Prior to installation all loose material within 3 ft of the seal will be removed from the roof, rib and floor. The mine entry seals will be made of solid concrete blocks (average minimum compressive strength of 1,800 lbf/in^2 tested in accordance with ASTM C140-70) and mortar (1 part cement, 3 parts sand and no more than 7 gallons of water per sack of cement) to form a wall two blocks thick.

Seals will be installed in the following manner:

a. The seal will be recessed at least 16 in. deep into the rib and 12 in. deep into the floor. No recess will be made into the roof. The blocks will be at least 6 in. high, except in the top course and 8 in. wide.

b. The blocks will be laid and mortared in a transverse pattern. In the bottom course, each block will be laid with its long axis parallel to the rib. The long axis in succeeding courses will be perpendicular to the long axis block in the preceding course. An interlaced pilaster will be constructed in the center. The seals will have a total thickness of 16 inches. Where conditions permit, the portal seals will be graded to conform with existing surface contours and seeded.

The opening in front of the wall will be filled with non-combustible material as above and the portal and entire exposed seam on the highwall will be covered with 6 to 8 ft of noncombustible material, graded, covered with suitable material and seeded. The portal fill material will be put in place with a load, haul, dump (LHD) unit to ensure proper backfilling. For illustration of a typical seal, see Figure 5-3. Temporary seals are discussed in R645-301-515.300.

Each mine entry which is temporarily inactive, but has a further projected useful service under the approved permit application, shall be protected by barricades or other covering devices, fenced, and posted with signs, to prevent access into the entry and to identify the hazardous nature of the opening, These devices shall be periodically inspected and maintained in good operation condition by the person who conducts the underground coal mining activities.

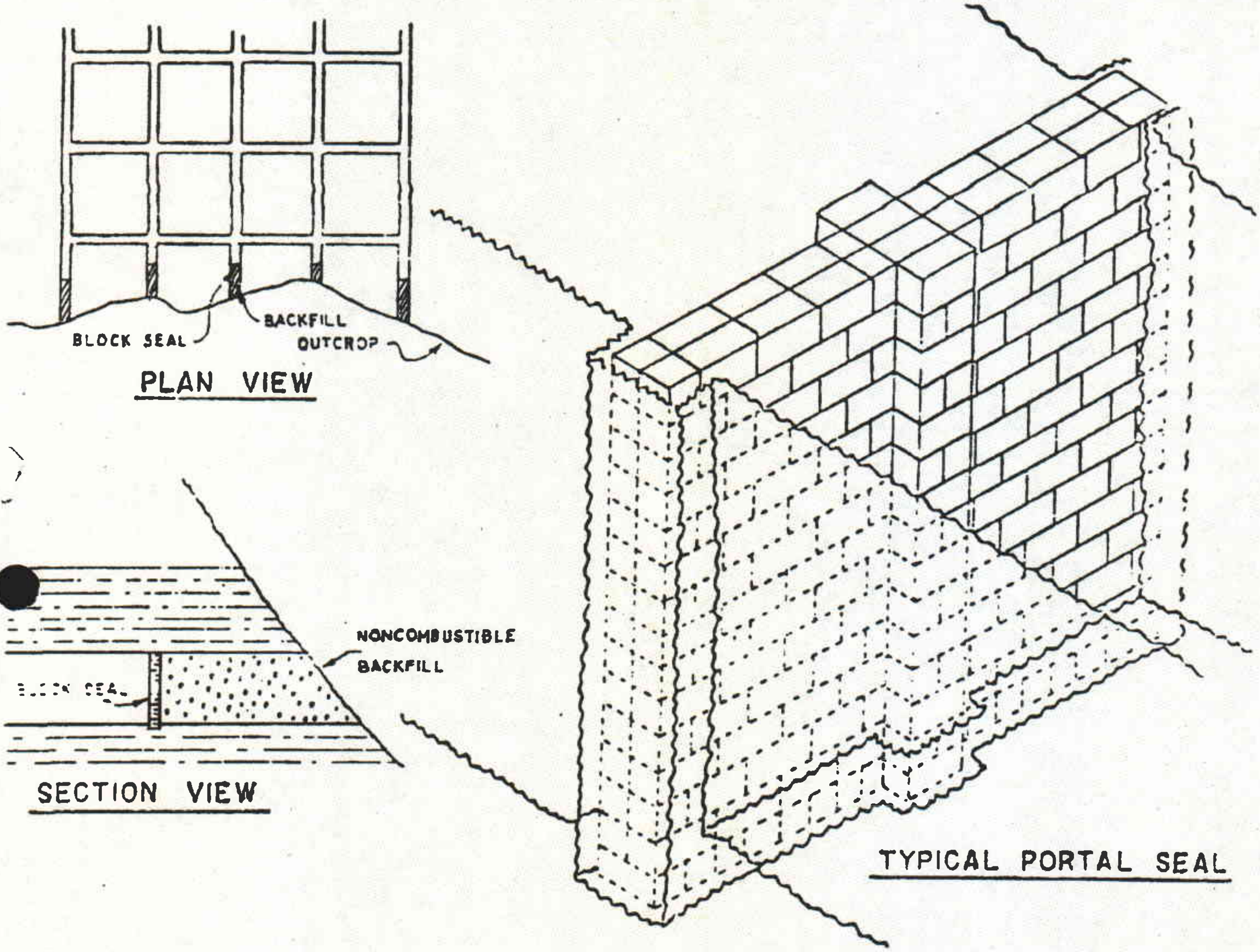


Figure 5-3 Typical Portal Seal

R645-301-530 Operation Design Criteria and Plans

531 General

General plans for sediment ponds are given in R645-301-528.400. Water impoundments are discussed in Chapter 7.

532 Sediment Control

Sediment control for each specific disturbed area is discussed and referenced under R645-301-526, and R645-301-732.

533 Impoundments

This is addressed in R645-301-528.400

534 Roads

This is addressed in R645-301-527. A reclamation timetable is given in R645-301-540.

535 Spoil

This is addressed in R645-301-514

536 Coal Mine Waste

This is addressed in R645-301-513, R645-301-514, and R645-301-528.

537 Regraded Slopes

537.100 Geotechnical Analysis

Slope stability analysis's are given in Appendix 5-G for the Tank Seam, Appendix 5-J for the WHR Blind Canyon Seam, Appendix 5-K for the WHR Tank Seam, and Appendix 5-L for the Mohrland area.

537.200 Settled and Revegetated Fills

Parts of the mine access roads will not be restored to its original contour, but will be left in place for post-mining land use. This is discussed in R645-301-242.

R645-301-540 Reclamation Plan

R645-301-541 General

541.100 Reclamation of All Areas

Upon completion of mining on the permit area, C.W. Mining will reclaim all disturbed surface areas as diligently and rapidly as possible, to restore the property to pre-mining and/or alternative post-mining uses. All reclaimed areas will be maintained during the liability period for at least 10 years.

The initial step in the reclamation plan is to seal all large-diameter openings by backfilling these openings with non-combustible material (earth & small rock), adjacent to the portals. The seals will be designed such that mine drainage, if any, will not enter surface water bodies. For a more detailed description of the sealing of openings, see R645-301-529, Sealing of mine Openings, Drill Holes, Wells, etc.

The next step in reclamation would be the removal of all surface structures, equipment and road blacktop. Once this has been accomplished, all solid waste generated in the abandonment operation will be collected and disposed of in an approved manner. Additional information concerning this aspect of the reclamation plan is present in R645-301-541.300 (Surface Structures), and R645-301-542.600 (Roads).

Proposed access roads to the mine portals will be reclaimed and revegetated. This will accomplish a dual purpose of controlling runoff and re-vegetating the hillsides with vegetation comparable to existing growth. Roads proposed to remain in place for post-mining use will be re-graded to the post-mining configuration and fitted with post-mining diversions. A detailed description of the roads is given in R645-301-542.600 (Roads).

Backfilling of the subterranean portion of the silos, holes and depressions will be the next reclamation activity, Once the backfilling is completed, the disturbed areas will be graded and recontoured. Topsoil materials will be redistributed as defined in Chapter 2 and then seeded or planted as described in Chapter 3, before the next growing season. A suitably permanent and diverse vegetation cover will be established on all affected areas of land.

The post-mining land uses will be grazing, recreation, wildlife and timber.

541.300 Removal of Structures

All surface facilities including support facilities will be removed and the areas reclaimed to prevent damage to fish, wildlife, and associated environmental values.

Mine Operational System Removal. Systems such as domestic water will be phased out and removed or buried. All structures, tanks and lines will be removed and properly disposed. Buried lines will be removed where feasible; otherwise, lines will be severed and left in place beneath a minimum of 4 ft of fill. Leaving lines in place will cause less disturbance than digging in some cases.

Excess Coal Spoil and Waste Disposal. Any excess coal spoil and waste material on-site at the time of reclamation will be placed and compacted against the cut slopes and covered with a minimum of 2 ft of non-toxic, non-acid material. If testing indicates that the material has acid or toxic forming potential it will be covered with a minimum of 4 ft of non-toxic, non-acid material. These materials will be placed in areas away from post mining drainage structures.

Solid (Non-Coal) Waste Disposal. The following are solid waste items expected during reclamation, and methods of disposal:

a. Concrete. All concrete (estimated at approximately 900 cu yds) will be broken up into pieces not to exceed 3 ft x 3 ft and placed against the highwalls mentioned above, and covered with at least 3 ft of backfill material. (This category also includes cinder blocks). Asphalt material will be taken to the Nielson Construction landfill;

b. Culverts and Pipe. All such material will be removed, and either salvaged or taken to the Nielson Construction Landfill in Emery County;

c. Scrap Metal. All scrap metal will be removed, and either salvaged or taken to the Construction Landfill;

d. Wood, paper, trash. All such items will be collected during the area cleanup during and after reclamation and then taken to the Nielson Construction Landfill for disposal. Wood items are identified as extraneous building material, timbers, block wedges and other mining material, and do not include any material occurring living or dead vegetation.

Note: The Nielson Construction Landfill is the designated disposal site for all non-coal solid waste items to be removed from the site.

R645-301-542 Narrative, Maps, and Plans

542.100 Timetable

The following schedule of reclamation is proposed to be initiated within 90 days (weather permitting) of final abandonment of the mining operation:

		Accumulated Time
a.	Seal Portals - 2 weeks	2 weeks
b.	Remove Structures - 34 weeks	36 weeks
c.	Soil Replacement and Ripping - 10 weeks	46 weeks
d.	Channel Restoration - 25 weeks	71 weeks
e.	Revegetation - 2 weeks	73 weeks

The above reclamation tasks can, therefore, be completed within 73 weeks, weather permitting, following the start of reclamation activities. Revegetation will be scheduled for completion in the fall of the final season of reclamation.

542.200 Backfilling and Regrading

General

The objective of the proposed backfilling, soil stabilizing, contouring and grading process is to achieve a reclaimed surface which will provide a variety of topographic features and enhance post-mining land use.

The steps to be taken in the backfill, soil stabilization, compaction, contouring and grading problems are described in the following subsections. Stability analysis of highwalls and backfilled areas are discussed in Appendix 5-H.

Backfilling operations, utilizing equipment such as rubber-tired scrapers, bulldozers, backhoes, front-end loaders and dump trucks, will be conducted in the portal and treatment facility areas. Holes or depressions will be filled when the mining operation is concluded. Compaction operations utilizing equipment such as sheeps-foot tampers will be conducted to stabilize all filled holes and depressions.

In general, the backfilling and grading operation will take place in the following manner:

a. All mining portals will be sealed and backfilled as previously described in R645-301-529.

b. Solid waste generated in the facilities removal will be collected and disposed of as identified in R645-301-541.300. See Appendix 5-D for toxic materials and handling.

In disturbed areas which contain coal fines from current operations and are not proposed to be regarded, the coal fines will be removed to pre-mining levels. Methods of removal will consist of either vacuuming (if justified by large quantities), or by washing down the area by high-pressure water hoses. The wash down procedure is particularly effective on rock and rocky slopes. All other extraneous debris from the operations will also be removed from the areas. Disposal of all materials will be as described in R645-301-529.

It should be noted that the existence of small to moderate amounts of coal fines has not been established as detrimental to either soils or vegetation; therefore, amounts less than the 50 pct figure cited above will not be removed.

c. A backhoe and dozer will work in conjunction to remove the outer edge of the recontoured operational benches and compact it against the cut slopes. This will be accomplished by the backhoe reaching over the edge of the bank approx 20 ft. and pulling the material back. The dozer will then push and compact this material from the cut slope outward to reach a bench slope of approx 1v:3h for drainage purposes and a maximum of 1v:2h on slopes outside of drainage areas. Culverts will be removed by excavating the material over the culvert, extracting the pipe, and backfilling the area.

d. This operation will start on the upper bench and work across the bench to the upper access road.

e. The backhoe and dozer will work in the same manner to eliminate the access road, working down to the lower pad. A typical cross-section of the reclaimed road cut is shown in Figure 3.6-2.

f. Steps d-f will be repeated from the Tipple Pad on down the canyon reshaping the mine yard and disturbed area to the configuration shown on Plates 5-6, Post-mining topography. The mine access road below the No. 3 Mine Access Road will be regraded and fitted with post-mining diversion structures as shown on Plates 5-6. Diversion designs are shown in Appendix 7-H. Asphalt road surfacing material from the scalehouse pad will be excavated and disposed of at the Nielson Construction Landfill in Emery County. All roads which are to be reclaimed will be closed to traffic during reclamation. The reclaimed road design will be the same as the operational design, and is shown on Plate 5-4.

g. As backfilling and grading is completed, operational areas will be scarified by gouging to a depth of approximately 8 in. with a trackhoe. All areas will be gouged to increase surface roughness. This will reduce compaction and prevent topsoil slippage, and improve soil retention and vegetation establishment in the gouges.

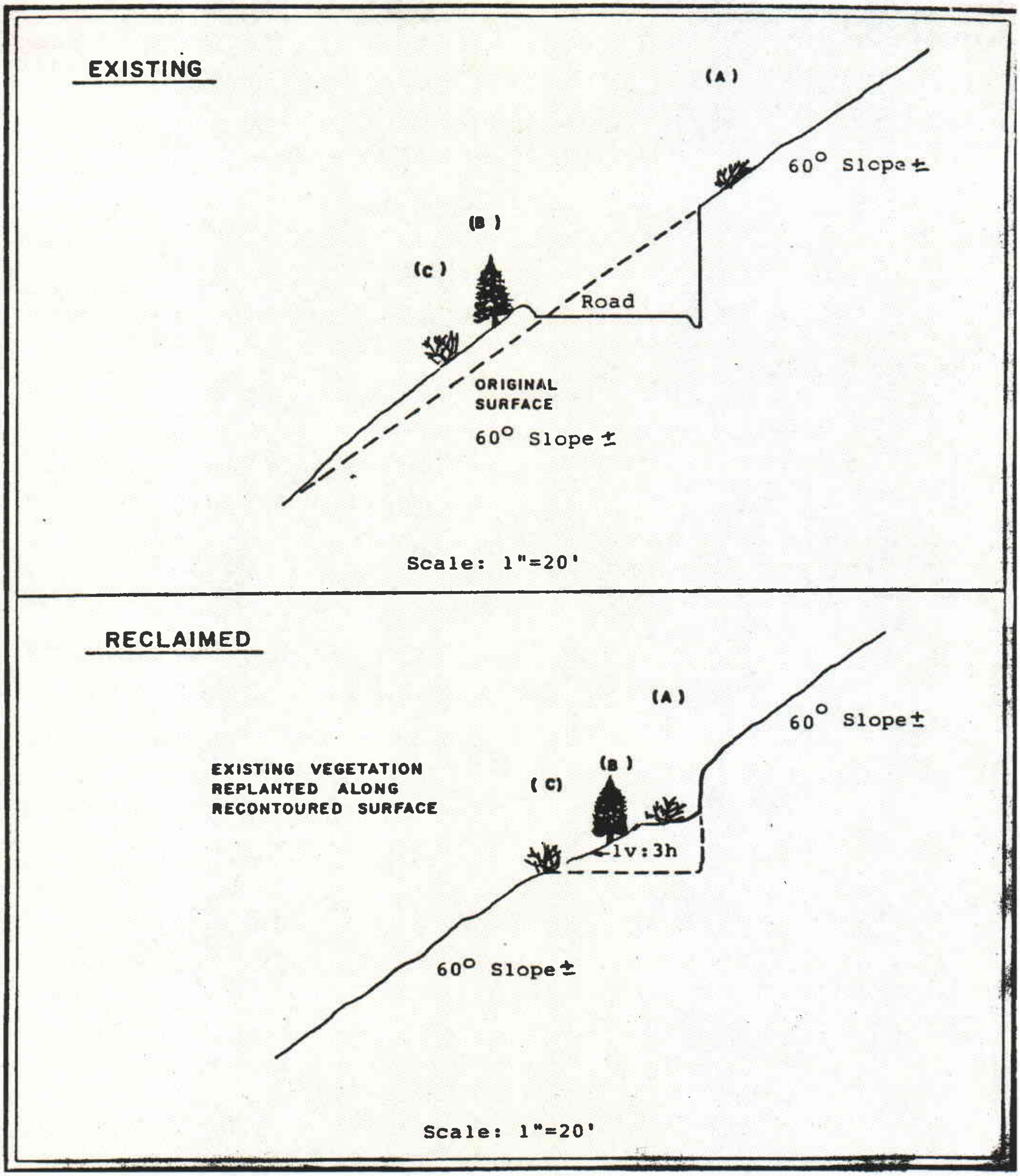


Figure 5-4 Typical Slope Reclamation

Tank Seam Access Road Reclamation

This section provides a detailed description of the backfilling and grading plans for the Tank Seam Access Road and Portal Area. Reclamation will be scheduled to allow revegetating of the area in conjunction with backfilling and grading.

As soon as the Tank Seam portals are sealed and all facilities removed, the borehole will be filled using the cuttings from initial construction of the borehole. These cuttings will be stored underground in the Blind Canyon Seam during operation. Backfilling will begin at the Tank Seam Portal Area. Backfilled material shall be placed by heavy equipment such as a front-end loader and track backhoe in lifts not exceeding 18 inches and compacted as described on page 5H-27. Lifts will be placed parallel to the slope of the road until the final lifts are reached which are wide enough for the equipment to drive on. Final lifts will be placed by a backhoe and compacted in a sloping manner as shown in Figure 5-5.

In areas where fill material is located on the outslope, fill material will be pulled back using a backhoe and the material incorporated into the lifts placed on the cut bench. Where large fill areas exist, the fill will be excavated in a reverse sequence from the placement described in Appendix 5-G. This will allow the fill to be removed and then reclamation to proceed from the bottom of the slopes toward the top, as the "pilot cut" is reclaimed.

Fill material will be excavated from around the large boulders, which were left in place during construction using a backhoe, and the approximate original contour of the slope will be restored. Channels will be restored in the large fill areas by duplicating the configuration shown in Appendix 7-H.

As the lifts are placed and fill material retrieved, topsoil will be redistributed on the outslope and upper lifts and spread using a backoe. Boulder sized rock fragments will be embedded into the surface in the upper lifts as described on page 5H-27 in order to obtain, at a minimum, a surface rock cover of 32.75 percent, which equals the reference area cover shown on page 5A-24.

Using the bucket of the backhoe, the surface will then be ripped and scarified, creating horizontal pockets approx. 8 to 12 inches deep to aid in water retention for revegetation. The bucket will also spread the topsoil at the top of the cut in a manner to blend the reclaimed slope with the natural slope above.

As backfilling progresses down the road, seed will be placed on the completed slopes behind backfilling by hand. Following seeding, erosion control matting similar to Excelsior S-2 or equivalent will be placed on the slopes by hand, and stapled to the reclaimed surfaces.

Reclamation will progress in this manner to the bottom of the Tank Seam Access Road.

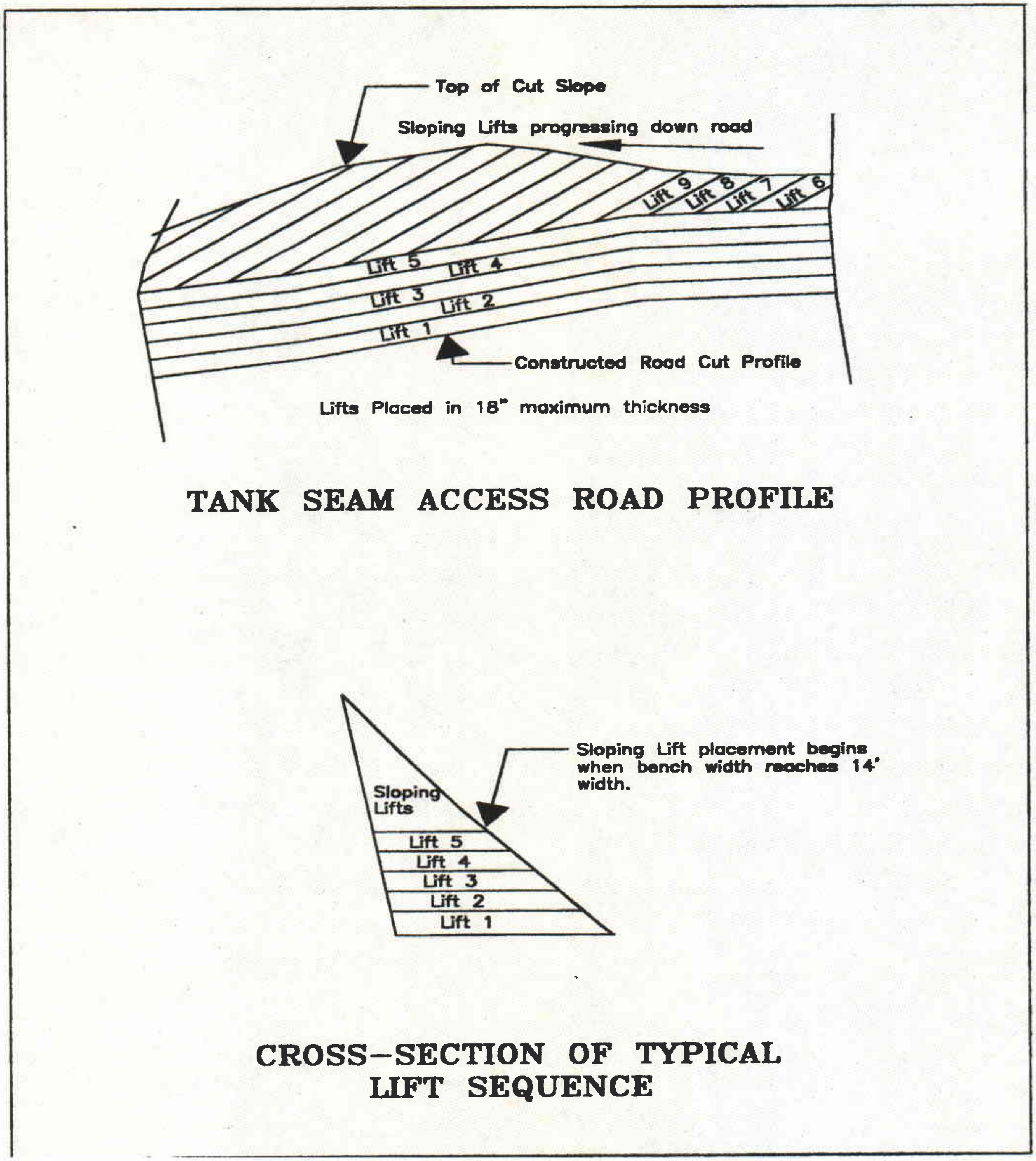


Figure 5-5 Typical Compaction Lift Sequence

Wild Horse Ridge Reclamation Plan

This section provides a detailed description of the backfilling and grading plans for the Wild Horse Ridge portal pads, conveyor roads and access roads. Reclamation will be scheduled to allow reseeding of the area immediately following backfilling and grading.

As soon as the portals are sealed and all facilities removed, backfilling will begin with the portal pads. ASCA Controls will be put in place in accordance with BTCA "2" controls, described in Appendix 7-K. Due to the remoteness of the area, Pond "D" will be removed during the initial backfilling and grading to avoid any future re-disturbance. A silt fence will first be placed at the base of the fill area across the drainage, as shown on Plate 5-6G. Fill material in the bottom of the canyon will then be excavated, including pond "D". Any highwalls created along the seam outcrops will be backfilled and the coal seams will be covered.

Material will then be placed along the portal pad access roads, backfilling the cut slope. Material shall be placed by heavy equipment such as a crawler dozer, front-end loader and track backhoe in lifts not to exceed 36 inches and compacted as described in Appendix 5-J. Material will first be excavated from the West side of the Blind Canyon Pad and the East side of the Tank Seam Pad (down-drainage side), maintaining access to the bottom of the canyon for topsoiling. As fill material is placed on the access roads, it will result in narrowing the road width, while backfilling the cut slope. Materials will be compacted in lifts not to exceed 24 inches in depth. Large diameter rocks will be incorporated into the outslope which is created by filling to aid in surface stability. This procedure will be followed until the majority of the cut is backfilled and the roads have been narrowed to "pilot cuts" which will still allow the equipment access to the areas. During the backfilling and grading process, culverts will be excavated and removed.

Topsoil will then be placed on the excavated areas of the pad and on the reclaimed slopes utilizing a trackhoe.

Any reclaimed slopes which will be out of reach of the trackhoe when backfilling is complete will receive topsoil during the backfilling process while the area is still within reach. As topsoil is placed, the surface will be roughened with deep gouges as described in Appendix 7-K.

The reclaimed channel, RC-9, will then be constructed in the drainage while the area is still accessible. The remaining fill material will then be excavated and temporarily placed along the portion of the access road which will remain in place for post-mining use and the post-mining roads will be reestablished. The excavated areas will be topsoiled in conjunction with the excavation. The "pilot cuts" will then be reclaimed in the same manner as the Tank Seam Access Road described above using the material previously placed along the remaining portions of the access road. When the backfilling is complete, silt fences will be staggered along the edge of the drainage as shown on Plate 5-6G for runoff control.

The conveyor access roads will be reclaimed in a similar manner. As all areas are reclaimed, the surface will be gouged using a trackhoe, creating pockets approximately 8-12 inches deep to aid in water retention for revegetation.

Since a cut slope existed along portions of this area prior to mining, there may not be enough material to completely eliminate the entire cut. In areas where cuts existed prior to mining, the material will be placed so as to backfill the cut to the extent possible. These areas are shown on Plates 5-6.

Slopes will be constructed in accordance with the slope stability recommendations presented in Appendix 5-J and Appendix 5-K. This will insure a slope stability factor greater than 1.3. Due to the limitation of a maximum slope of 2H:1V, some portions of the road and pad cuts will be left during reclamation. The locations of the cuts are shown on Plates 5-6F and 5-6G. The heights of these cuts are summarized on page 5J-64. These cuts are also shown on the cross-sections in the post-mining slope stability analysis in Appendix 5-J and Appendix 5-K.

Following regrading, seed and fertilizer will be placed on the completed slopes by hand. Following seeding, erosion control matting similar to Excelsior-S-2 or equivalent will be placed on slopes constructed at or near 2h:1v. The erosion control matting will be installed in strict conformance with the manufacturer's instructions. Following the placement of the matting, the temporary silt fences placed for runoff control during reclamation will be removed.

542.300 Final Surface Configuration Maps and Cross Sections

Plates 5-6 show the final surface configurations after reclamation. Cross sections on shown on the maps are found in Appendixes 5-I, 5-J, and 5-K.

542.400 Removal of Temporary Structures

See R645-301-240, R645-301-340, R645-301-412, R645-301-540, and R645-301-740.

542-500 Timetable for Removal of Water Impoundments

See R645-301-542.100

542.600 Roads

See R645-301-240.

542.700 Final Abandonment of Mine Openings and Disposal Areas

See R645-301-529 and R645-301-738.

542.800 Reclamation Costs

See R645-301-830.100 and R645-301-830.300

R645-301-550 Reclamation Design Criteria and Plans

R645-301-551 Casing and Sealing of Underground Openings

The Bear Canyon Mine complex has eight existing portals.

The Bear Canyon #3 and #4 mines have a total of six portals (Plate 5-1A and 5-1C), all located in Bear Canyon. The #4 Mine has an auxiliary portal described in Appendix 5P. All portals will be reclaimed as described in sections R645-301-529, and R645-301-738.

R645-301-552 Permanent Features

No response required.

R645-301-553 Backfilling and Grading

553.100 Disturbed Area

See R645-301-240, R645-301-412.100, and R945-301-540.

553.200 Spoil and Waste

See R645-301-513, R645-301-514, and R645-301-525.

553.250 Refuse Pile

Not applicable.

553.260 Disposal of Coal Processing Waste

See R645-301-513.

553.300 Exposed Coal Seams

See R645-301-731.100 and R645-301-731.300

553.400 Cut and Fill Terraces

No cut and fill terraces are proposed.

553.500 Previously Mine Areas

See R645-301-540

553.600 Approximate Original Contour

See R534-301-540

553.700 Backfilling and Grading Thin Overburden

Not applicable.

553.800 Backfilling and Grading Thick Overburden

Not applicable

R645-301-560 Performance Standard

Coal mining and reclamation operations will be conducted in accordance with the approved permit and requirements of R645-301-511 through R645-301-553.

Appendix 5-A
Existing Structures

EXISTING STRUCTURES

Table 5A-1 lists each structure and construction dates. Reclamation is expected in 2012.

Table 5A-1 Existing Structures

Existing Structure	Construction Dates Starting	Construction Dates Completion	Photo #
Sales/Receiving/Scale Office/Caretaker Dwelling	6/84	10/87	1
Fuel Tanks	10/83	6/84	2
Truck Loading Facility	9/82	4/83	3
Oil Slack Loading Facility	4/83	7/83	3
Storage & Stacking Facility	6/80	4/84	3
Conveyor Structures	3/80	6/80	3
~~Added~~ Machine Shop	11/89	12/89	5
Shop	10/83	9/84	4
Coal Processing Facility	4/80	12/85	6
Lump Coal Facility	10/83	12/85	6
Non-Coal Storage Yard	3/80	9/84	7
~~Transformer Sub-Station~~	~~4/80~~	~~6/80~~	~~8~~
~~Cross Conveyor~~	~~7/89~~	~~9/89~~	~~9~~
WHR Tank Seam Fan	7/4/01 ~~proposed~~	12/31/05	10
~~Coal Storage Bin~~	~~4/87~~	~~10/87~~	~~11~~
Powder Magazine	9/82	containerized	
Water Tanks & System	8/82	11/82	13
~~Mine Fan~~	~~9/82~~	~~11/82~~	~~14~~
Lump Coal Storage Pad	8/92	10/92	15
Equipment Wash Pad	8/92	10/92	16
Shower House	5/93	7/94	17
Antifreeze Storage Tank	12/93	1/94	18
WHR Blind Canyon Seam Fan	7/4/01	12/31/05	19
Wild Horse Ridge Conveyor Belt	7/4/01	12/31/05	9
WHR Substation	7/4/01	12/31/05	12
~~WHR Retaining Wall~~	~~7/4/01~~	~~12/31/05~~	
WHR ~~Water &~~ Fuel Tanks	7/4/01	12/31/05	14
WHR Coal Storage Bin	7/4/01	12/31/05	
Power Lines	7/4/01	12/31/05	
Water Lines	7/4/01	12/31/05	
~~Portable Fan~~	~~7/4/01~~	~~12/31/05~~	~~21~~
Fuel Containment Enclosure	7/4/01	12/31/05	
~~Tank Seam Borehole Structure~~	~~7/4/01~~	~~12/31/05~~	~~20~~
Mine Portals	-	-	-

The location of each of the listed structures is shown on Plate 5-2, Surface Facilities. Co-Op has sought interim approval for each structure in the course of construction. Hydrologic safeguards have been implemented and are described in Chapter 7 and shown on Plates 7-1A through 7-1G. Topsoil has been removed and stored as shown in Chapter 8, and interim revegetation has been completed where the earthwork is at final grade. Health and safety standards are implemented as per MSHA standards.

The support facilities will be operated in accordance with the Bear Canyon Coal Mining and Reclamation Permit.

All of the structures are to be reclaimed during reclamation as described in Chapter 8. In order to consolidate all previous plan submittals, current photographs were taken 5/90 and are attached herein. A brief description of each facility follows under "Facility Description".

FACILITY DESCRIPTIONS

1. <u>Sales/Receiving/Scales Office/Caretaker Dwelling</u>. This structure contains the parts warehouse, parts receiving, scale office, mine offices, and the Caretaker Dwelling. See Photo #1.

2. <u>Fuel Tanks</u>. There are three 10,000 gal. fuel storage tanks installed at the downslope of the shop area. These tanks are contained within a natural berm of

the slope with the only access by way of the disturbed drainage ditch leading directly to the sediment pond. The pond is designed to contain any spillage which could foreseeably occur. The area will be posted " No Smoking " and fire extinguishers are in place. All MSHA safety standards will be adhered to. See Photo #2.

3. Truck Loading Facility. The truck loadout is a conveyor system designed to load tractor-trailer trucks from any of the storage pile areas. It is electrically manipulated so as to minimize spillage. The area is cleaned of spilled coal as needed. All runoff is routed to the Sediment Pond "A". See Photo #3.

4. Oil/Dry Slack Loading Facility. The slack loadout is designed to handle oiled and non-oiled stoker coal, primarily for non-commercial clients. It includes a 20,000 ton storage bin with an electrically operated auger to load small tonnages. The bin is fed by a hopper and conveyor, which is loaded with a front end loader. See photo #3.

5. Coal Storage Area and Stacking Facility. The coal storage yard is equipped with a system of conveyors wherein coal can be segregated according to size. This storage is of a short-term nature; the piles are constantly being consumed and replenished. The area also contains two 6,000 gal oil storage tanks, which are used to store oil for stoker coal. All run-off is controlled, and passes through the Sediment Pond "A". See Photo #3.

6. Conveyor Structures. These conveyors are the route by which the coal exits the mine to the storage piles and loadouts. See Photo #3.

7. Shop/Machine Shop. The shop building is for servicing of both underground and surface equipment. The building is heated with a coal furnace and is equipped with standard heavy equipment handling implements such as winches, welders, etc. See Photo #4. A new machine shop (30 ft by 40 ft) was added in 1989 to better facilitate mine related repairs. See Photo #5.

8. Coal Processing Facility. This facility is primarily a coal sizing site where the various sizes of coal can be made and then stacked in the designed locations. Runoff from this area is routed to the Sediment Pond "A". See Photo #6.

9. Lump Coal Facility. This structure consists of a storage bin and loading conveyor. See Photo #6.

10. Non-Coal Storage Yard. This area is utilized for all material which is in storage on the property with projected use and/or salvage value. Runoff from this area is routed to the Sediment Pond "A". See Photo #7.

11. Transformer Substation. ~~This facility supplies electrical power to both surface and underground facilities. A fence is maintained around the structure, and the area complies with MSHA health and safety standards.~~ This structure has been reclaimed See Photo #8.

~~Cross Conveyor. This belt conveys the coal from the Blind Canyon Seam to the coal storage bin. See Photo #9.12.~~

12~~13~~. WHR Tank Seam Fan. The Wild Horse Ridge Tank Seam Fan (Shown in Photo #10) was the old Blind Canyon Fan and is MSHA approved. All safety guards are maintained and in place.

13~~14~~. Coal Storage Bin. ~~This structure consists of a 20 ft X 20 ft surge bin to receive coal from the underground conveyors prior to traveling to the crusher. The structure is completely enclosed in order to control coal fines.~~ This structure has been reclamed. See Photo #11.

14~~15~~. Powder/Cap Magazines. This structure consists of a fire proof storage housing. See Photo . These structures comply with all requirements for Type 2 magazines as described in Sec. 1102, United States Code, Chapter 40, Subpart K - Storage, Section 55.208.

15~~16~~. Water Tanks. These surge tanks are a part of the culinary water supply system. See Photo #13.

~~17. Mine Fan. The mine ventilation fans include the Bear Canyon Fan (shown in Photo #14). The fan is MSHA approved, and all safety guards are maintained in place.~~

16~~18~~. Lump Coal Storage Pad. This structure consists of a concrete pad and concrete retaining walls. See Photo #15.

17~~19~~. Equipment Wash Pad. This structure consists of a concrete pad with a grease and oil trap. The grease and oil trap will be cleaned quarterly to prevent material buil-up. Material will be disposed of in the Emery County approved landfill. See photo #16.

18~~20~~. Shower House. This structure consists of a two story masonry block building that houses employee showers, training classrooms and offices. See Photo #17. The waste disposal system is discussed in Appendix 5-N.

19~~21~~. Antifreeze Storage Tank. This consists of 2,000 gal storage tank. Antifreeze solution is used to spray truck hoppers during periods of cold weather to prevent coal from freezing in transit. The tank is enclosed by a metal structure to hold the entire tank capacity in the event of a spillage. See photo #18.

2022. WHR Blind Canyon Seam Fan. The WHR Blind Canyon Seam Ventilation Fan came from the old Tank Seam mine and is MSHA approved, and all safety guards are maintained in place. See photo #19.

2123. Wild Horse Ridge Conveyor Belt. This structure transports coal from the Bear Canyon No. 3 Mine to the tipple facilities.

2224. WHR Substation. This facility supplies electrical power to the surface and underground facilities operating in Wild Horse Ridge.

~~25. WHR Retaining Wall. The wall is 30 feet tall and 300 feet long and was installed to allow for the widening of the upper Tank Seam access road.~~

23~~26.~~ WHR ~~Water and~~ Fuel Tanks. These tanks are used to store ~~water and~~ diesel fuel. ~~The water is used for mining operations in the Bear Canyon No. 3 Mine~~. The fuel tank is used to refuel diesel equipment used in the operation of the No. 3 Mine. ~~The water tank came from the old T.S. mine.~~

24~~27.~~ WHR Coal Storage Bin. This structure is used for coal surge capacity from the Bear Canyon No. 3 and 4 Mines. It consists of a metal storage silo approximately 30' diameter.

25~~28~~. Power Lines. Power is supplied to the mine facilities through high voltage power lines. The line pole locations are shown on Plates 5-2A through 5-2G.

26~~29~~. Water Lines. Water is supplied to the mine facilities with the use of a piping network. Water lines are shown on Plates 7-1A through 7-1G.

~~30. Portable Fan. This fan was added to the Bear Canyon #1 Mine to act as an auxiliary fan to the Bear Canyon #1 Mine Fan.~~

27~~31~~. Fuel Containment Enclosure. This structure is designed to contain material from the storage tanks if they should rupture. There are three tanks located within the enclosure, two 11,500 gal tanks and one 17,500 gal. The enclosure will consist of the base and 5 walls each 2'6" high enclosing an area of 1,500 ft^2. The structure will hold over 22,000 gal. Calculations are shown below.

V_{req} = 17,500 gal. * 1.1 = 19,250 gal = 2,600 ft^3

Enclosed Area = 1624 ft^2.

11,500 gal. Tank area = p* $(7\ ft)^2$ = 155 ft^2

Containment Area = 1624 – (2*155) = 1,314 ft^2.

Wall height = 2,600ft^3 1,314 ft^2 = 2 feet + 4 inches freeboard

Actual wall height = 2 feet 4 inches.

Spill material will be drained out the bottom through a pipe with a locking valve and transported and disposed of in accordance with all state and federal regulations. The enclosure will be checked weekly and drained of standing water if needed. Details of the design, maintenance, and spill disposal can be found in the C.W. Mining SPCC plan.

~~32. Tank Seam Borehole Structure. This metal structure fully encloses the borehole and conveyor, which conveys coal from the Tank Seam Mine to the Blind Canyon Seam Mine. See photo #20.~~

28~~33~~. Portals. ~~Bear Canyon Mine Complex has seven existing portals, and one proposed Portal. The Blind Canyon Seam (Plate 3-4A) has two fans, one belt, and two intake portals. The first fan portal is in Bear Canyon near the upper storage pad and the second is in the Blind Canyon. The belt portal pad is shown on plate 3-6. One intake portal is located in the main portal area, and one in Blind Canyon (Appendix 3-I). Three accidental breakouts also exist in Blind Canyon, making a total of 5 openings in the Blind Canyon Seam on the Blind Canyon side. Four of these have been reclaimed in the manner described in (Appendix 3-I). The remaining two have been permanently sealed and will be backfilled during final reclamation. There are two portals in the Hiawatha Seam (Plate 3-4B): a belt and and intake portal. Permanent seals have been places over there portals backfilling will take place during final reclamation.~~

~~The bear canyon #2 mine, has three portals (Plate 3-4C), that have been reclaimed.~~

The Bear Canyon #3 and #4 Mines, in Wild Horse Ridge, will have a total of six portals (Plate 3-4A and 3-4C), all located in Bear Canyon.

A Summary of the Portals are as follows:

		Existing	~~Proposed~~
Blind Canyon Seam	~~- Bear Canyon~~	3~~4~~	
	~~- Blind Canyon~~		
~~Hiawatha Seam~~	~~-~~	~~2~~	
Tank Seam	-	3	
	Total	6 ~~9~~	

Photo #1 Sales Receiving - Scale Office

Photo #2 Fuel Tanks

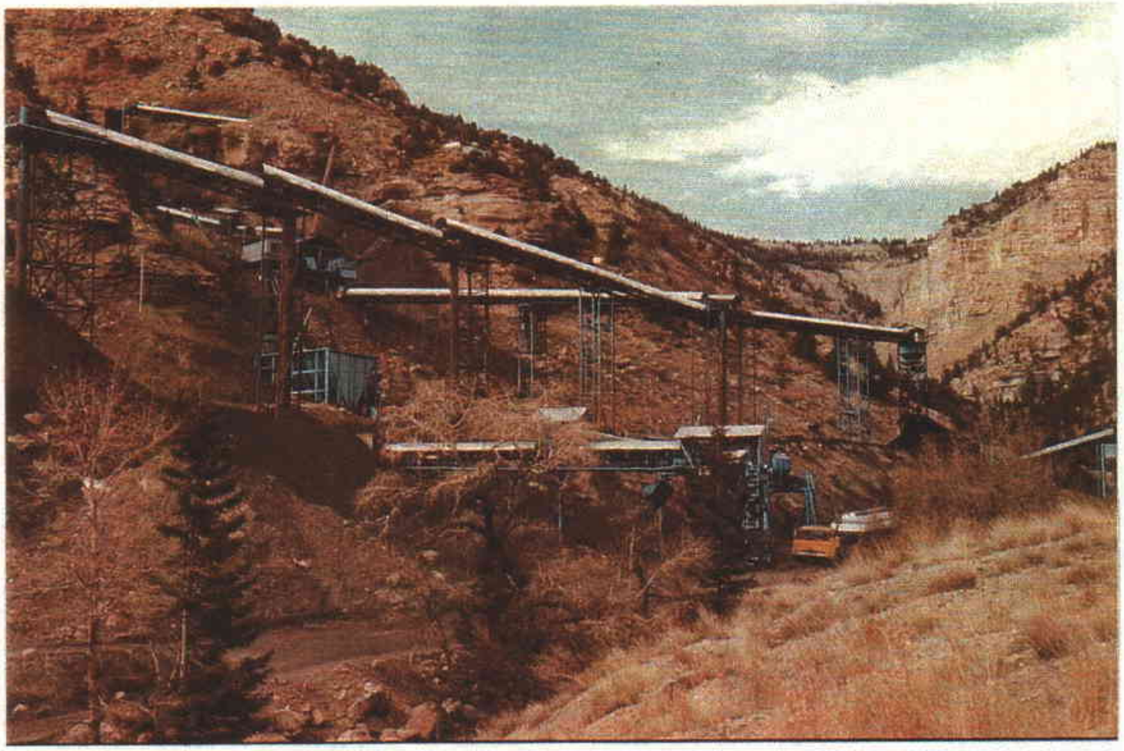


Photo #3 Truck Loading Facility, Oil Slack Loading Facility, Storage & Stacking Facility

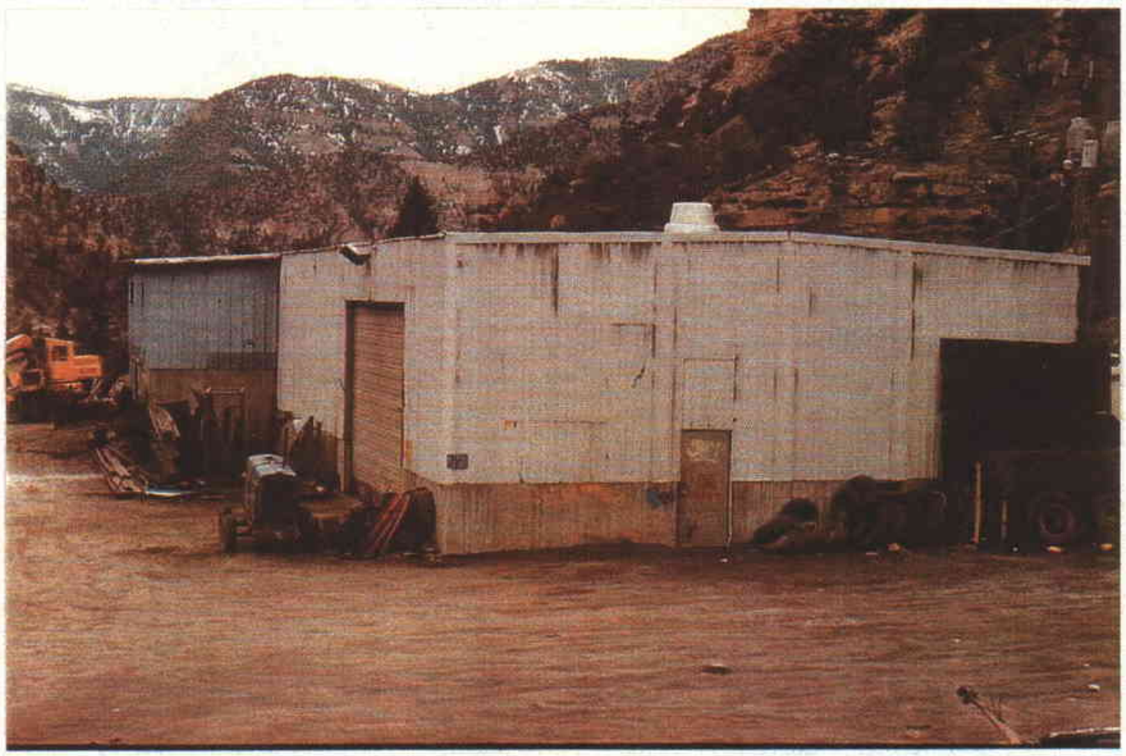


Photo #4 Shop

Photo #5 Machine Shop

Photo #6 Coal Processing Facility, Lump Coal Facility

Photo #7 Non-Coal Storage Yard

This structure has been reclaimed
Photo #8 Transformer Sub-Station

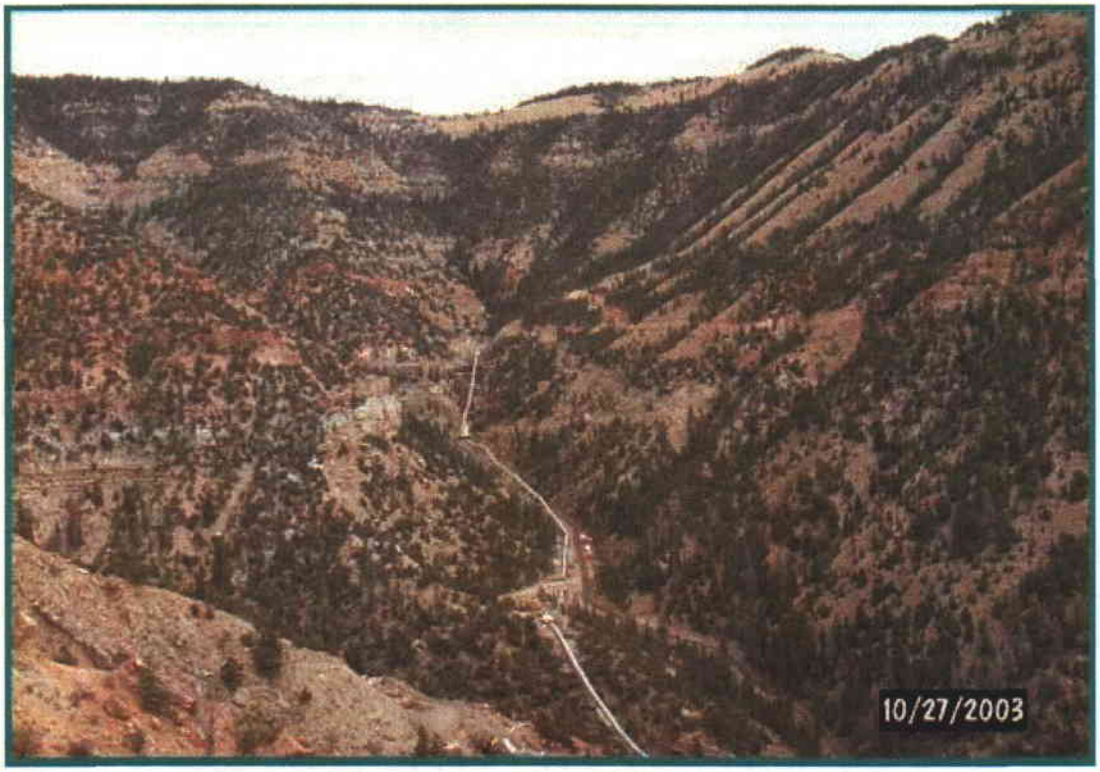


Photo #9 ~~Cross~~ Wild Horse Ridge Conveyor Belt

Photo #10 WHR Tank Seam Fan

This structure has been reclaimed.
Photo #11 Coal Storage Bin

Photo #12 WHR Substation

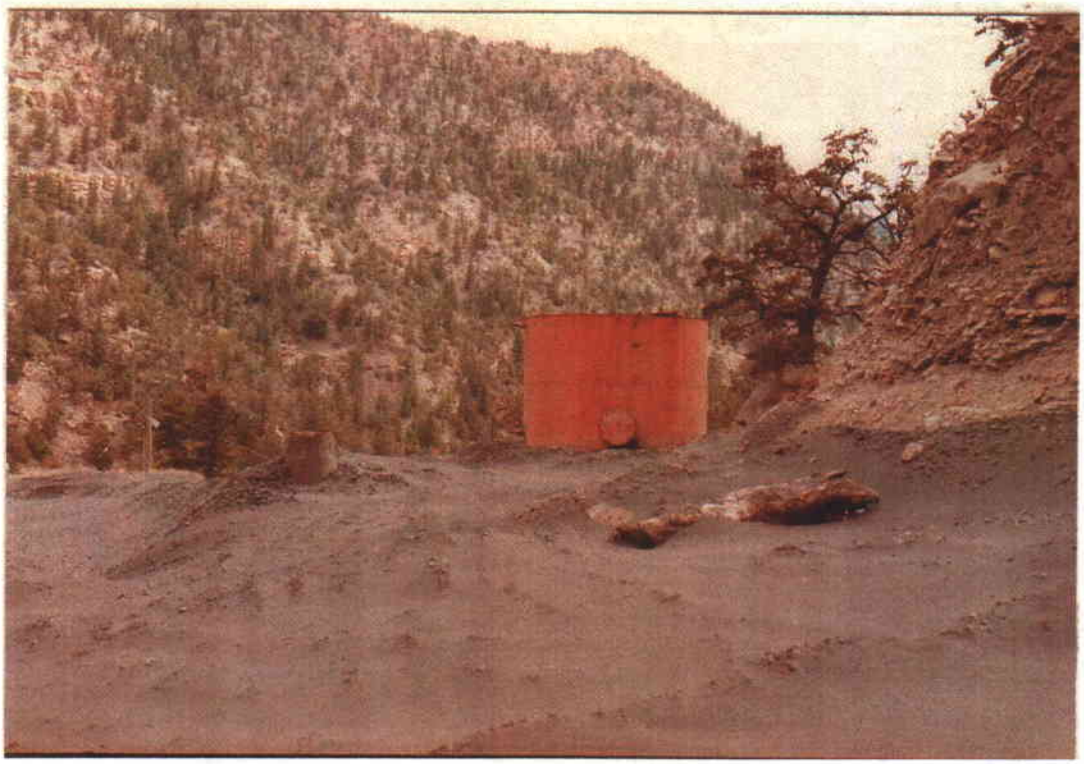


Photo #13 Water Tank

Photo #14 WHR Fuel Tank~~Mine Fan~~

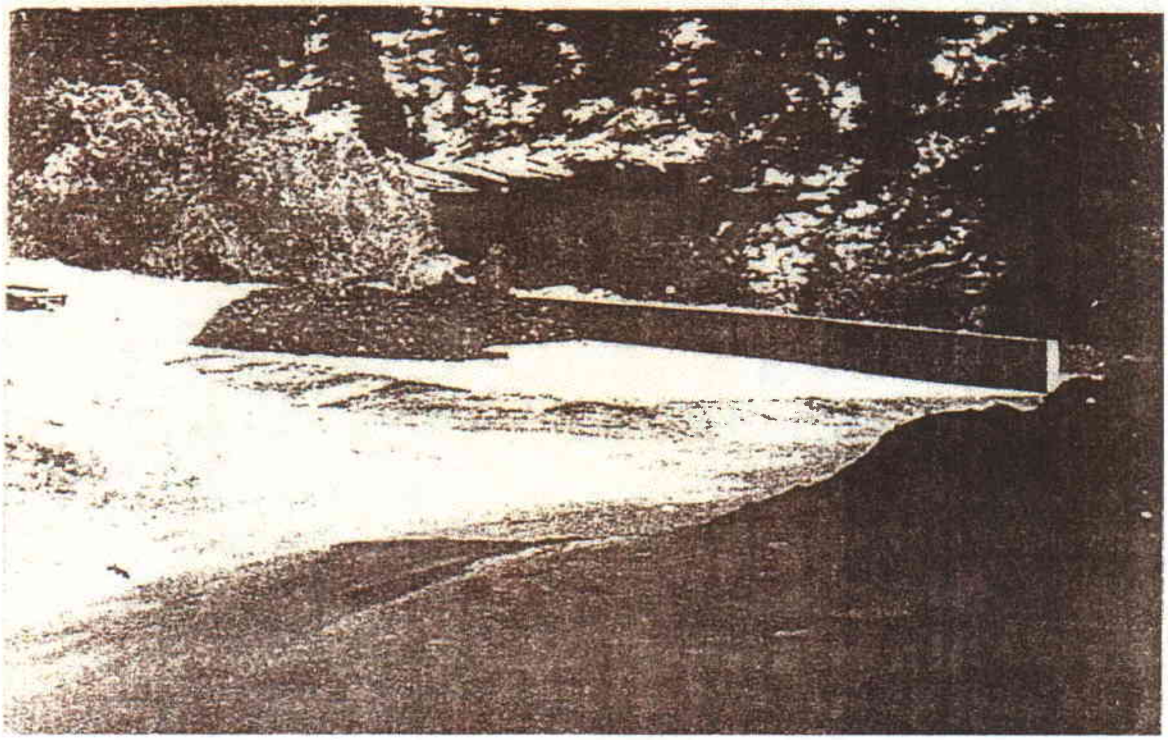


Photo #15 Lump Coal Storage Pad

Photo #16 Equipment Wash Pad

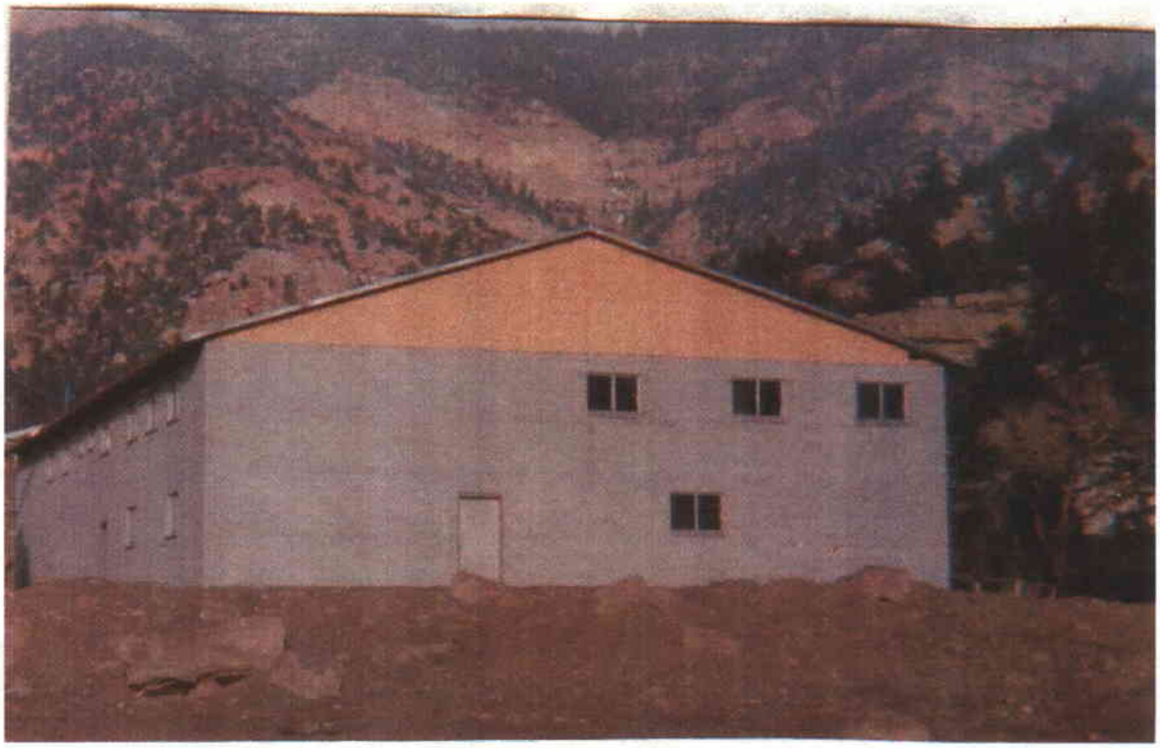


Photo #17 Shower House

Photo #18 Antifreeze Storage Tank

Photo #19 WHR Blind Canyon Seam Fan

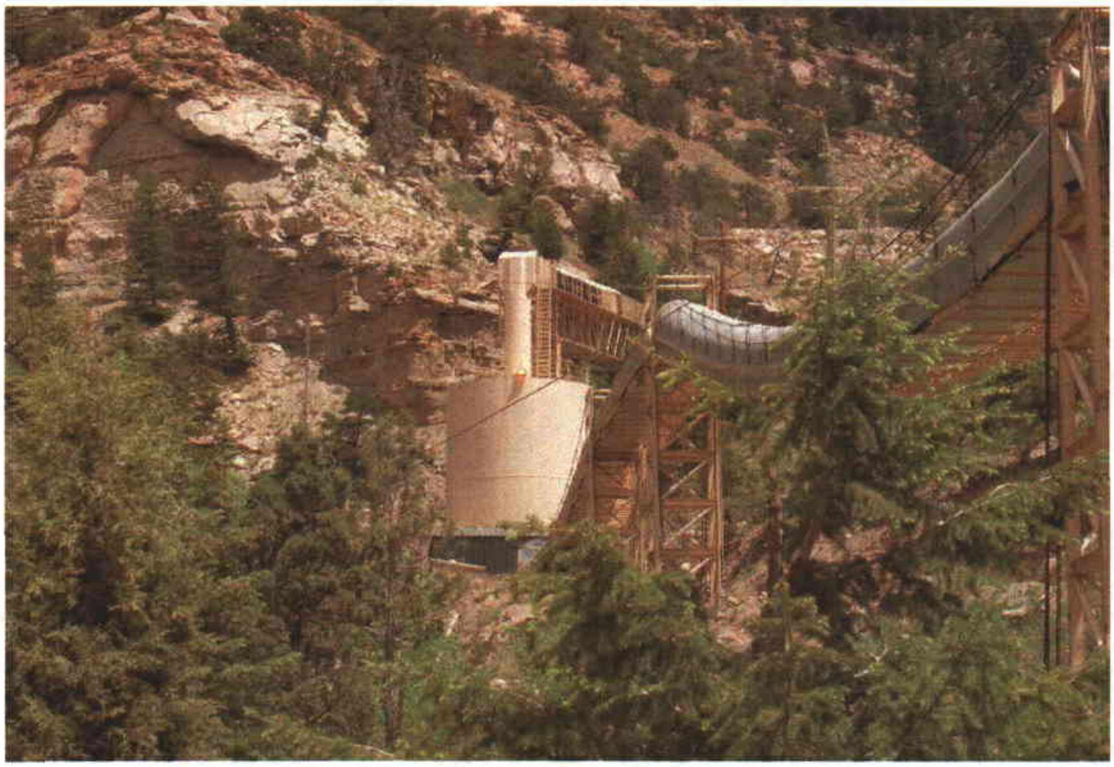


Photo #20 WHR Coal Storage Bin ~~Tank Seam Borehole Structure~~

Appendix 5-B

Culinary Water System

Culinary Water System

Co-Op, on September 1983, applied for a change of Point of Division place and nature of use of water Right A-35836 (93-1067). The application was approved on September 28, 1984. The purpose of the application and the change was to facilitate a new mine located in Bear Canyon SE1/4,SE1/4, Sec. 22, NE1/4, NE1/4, Sec. 27 T16S, R7E, SLBM.

On June 10, 1985, and application for a change in nature of use was submitted to the Division of Water Rights to incorporate culinary (domestic use). Anticipated usage of the system is a potential of 2 families, and as many as 40 miners on 3 different shifts.

Water is trapped in a portion of the old works of The Bear Canyon Mine. This mine was abandoned in the 1920's. The resulting of abandonment created a sump which is recharged through a number of seeps associated with regional faulting. A 1 ½ in. PVC schedule 40 pipe was installed to gravity feed a 10,00 gal water storage tank located at the fan site. A distance of approximately 360 ft transversing the existing mine. This pipe is buried on all working areas of the mine to a depth of 4 ft.

Laboratory analysis of the water indicates that it is of a suitable quality, and a sample sent to the Department of Health to determine bacterial content, indicated "O" count.

The culinary water line is shown on Plate 2-4. Copies of the system as approved by the Utah State Division of Health, Safe drinking water Committee are included in the following pages of this appendix.

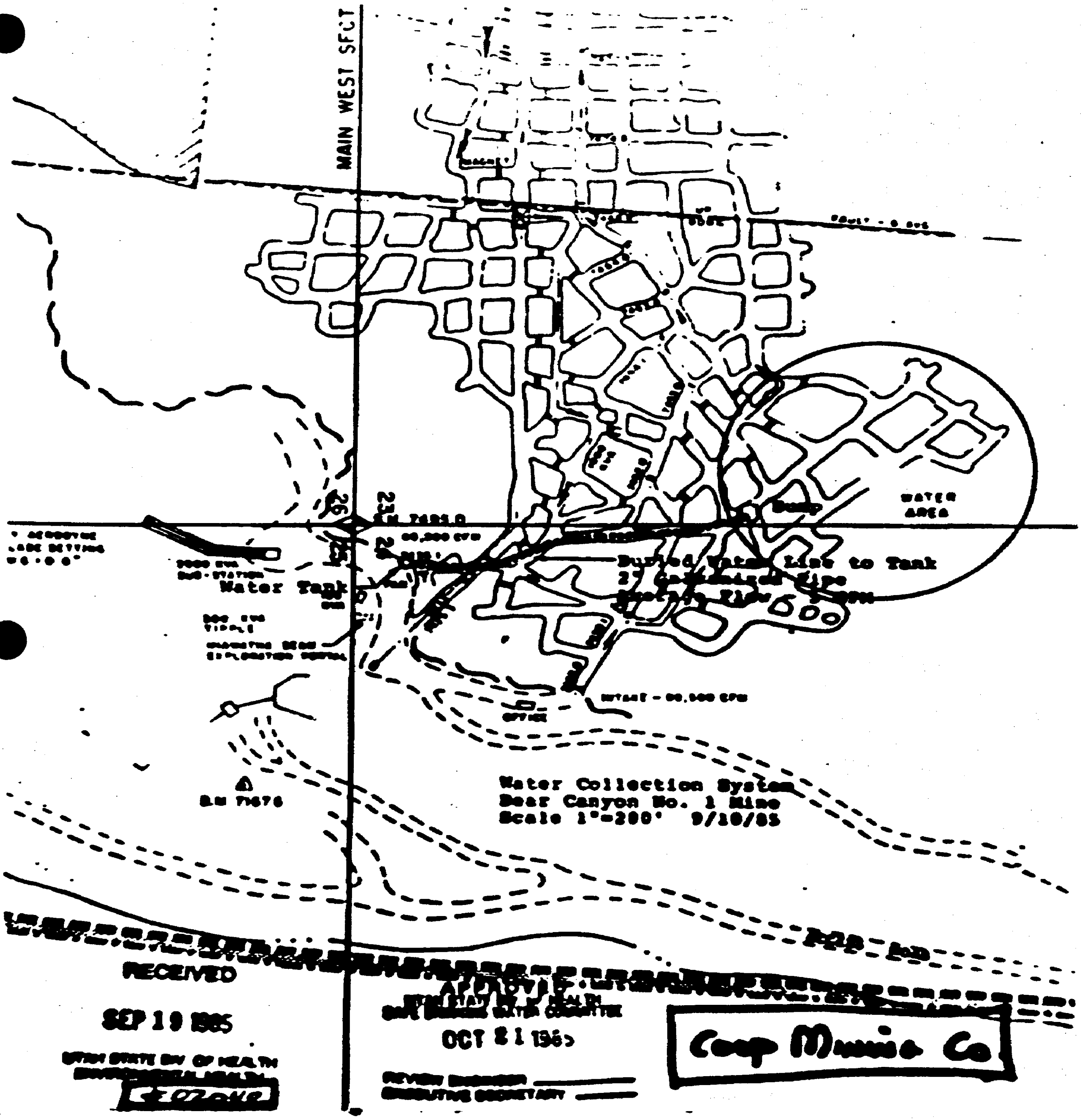


Figure 5B-1 Water Collection System, Bear Canyon Mine

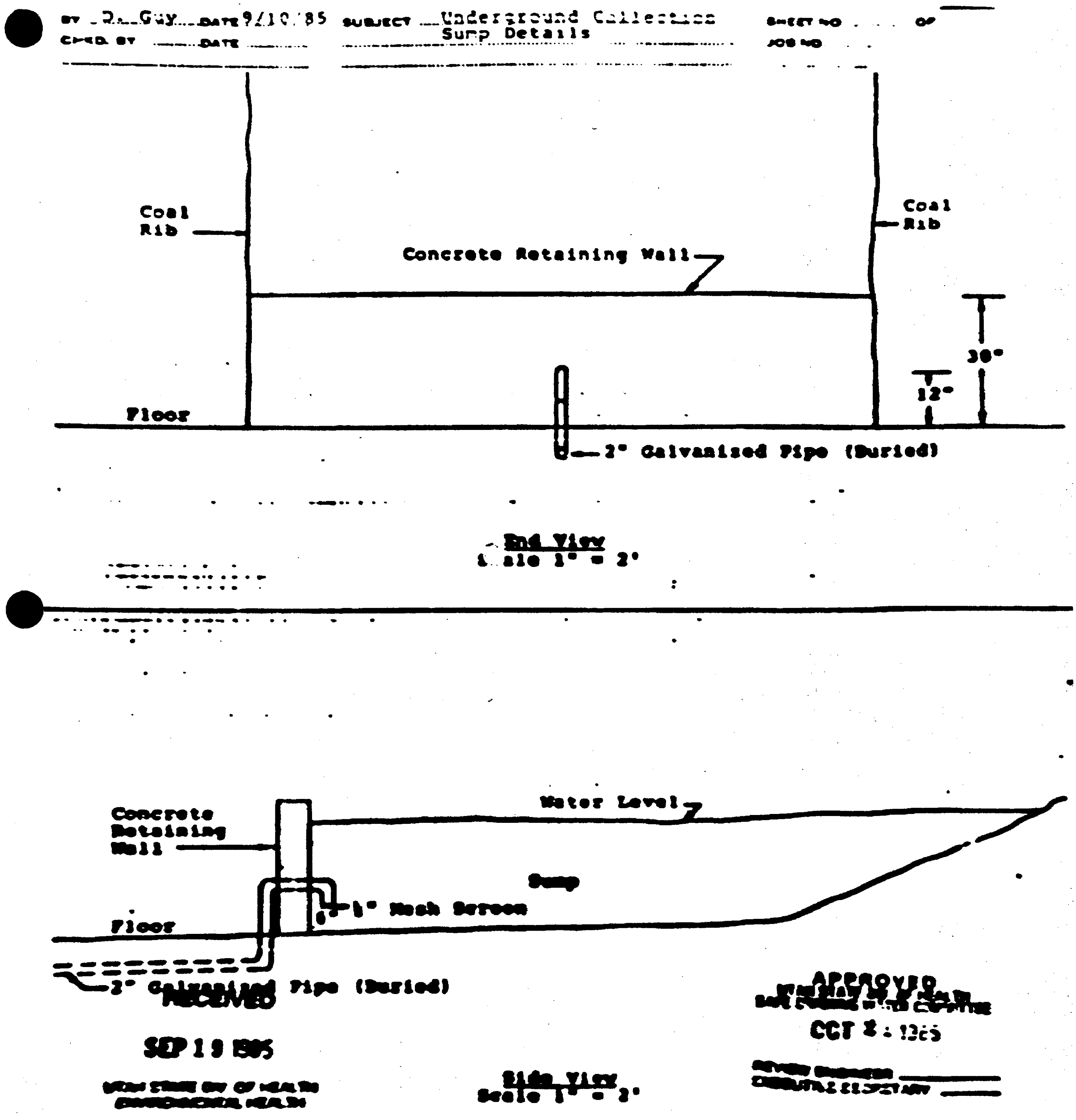
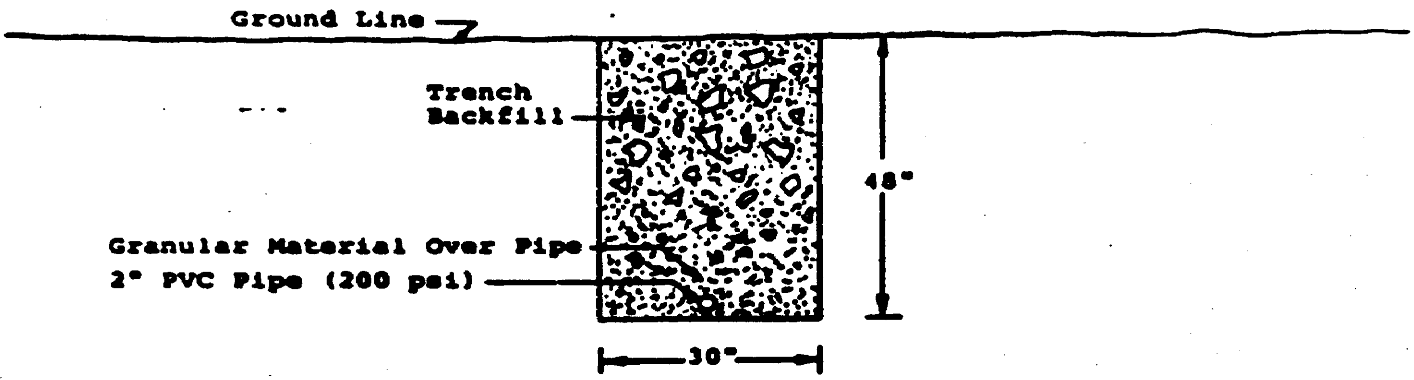


Figure 5B-2 Underground Collection Sump Details

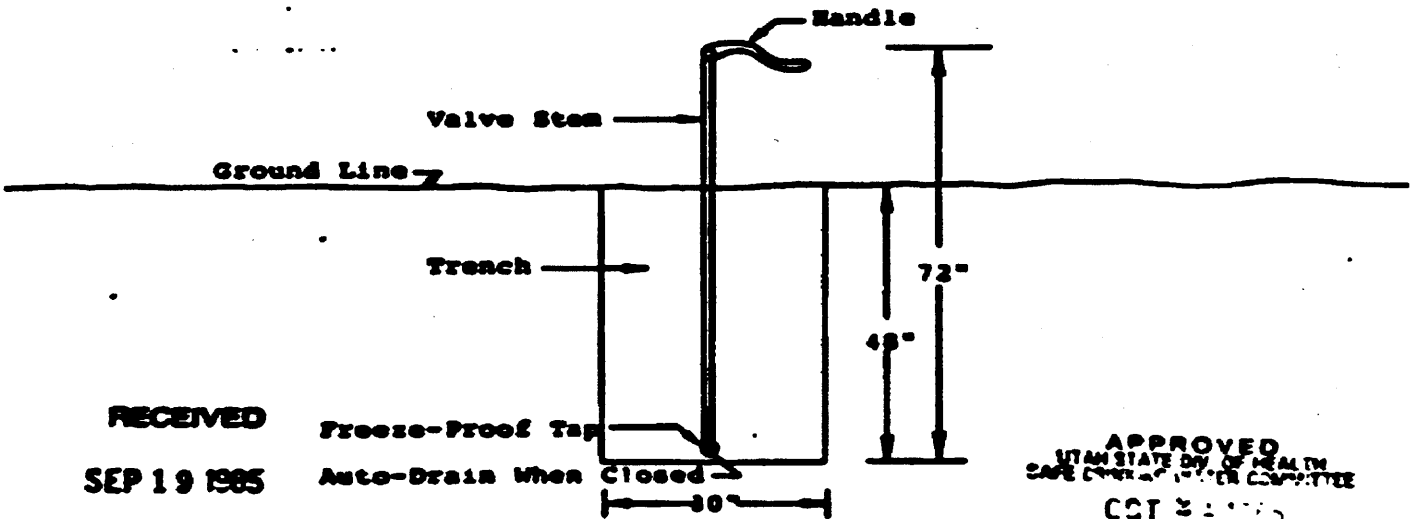
BY D. Guy DATE 9/10/85 SUBJECT Typical Section of Buried Water Line and Freeze-Proof Valve SHEET NO. ...... OF ......

CHKD. BY ...... DATE ...... JOB NO. ......

Scale 1" = 2'



End View of Trench and Pipe



Freeze-Proof Valve

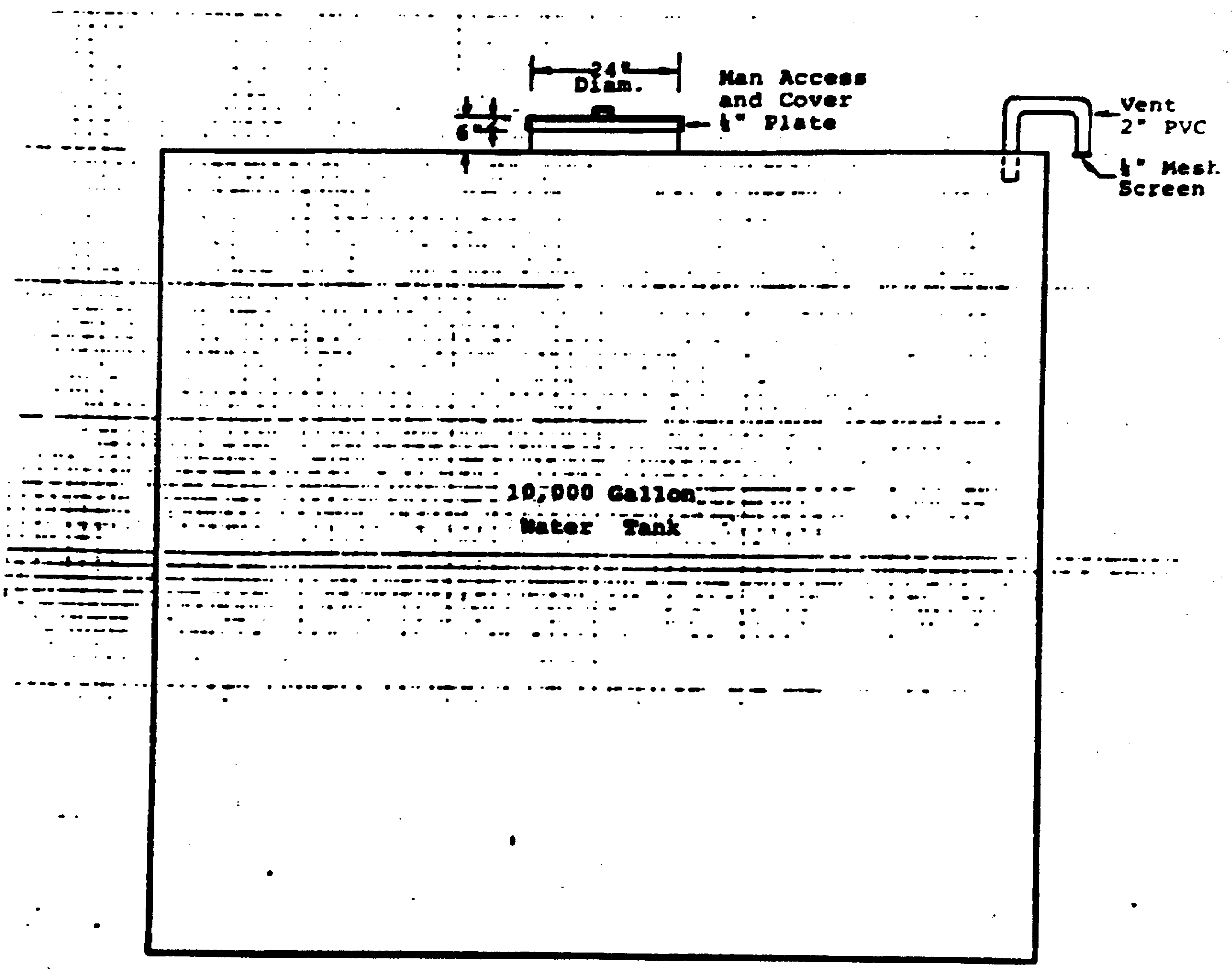
RECEIVED
SEP 19 1985
UTAH STATE DIV OF HEALTH
ENVIRONMENTAL HEALTH

APPROVED
UTAH STATE DIV. OF HEALTH
SAFE DRINKING WATER COMMITTEE
OCT
REVIEW ENGINEER ______
EXECUTIVE SECRETARY ______

Figure 5B-3 Typical Section of Burried Water Line and Freeze-Proof Valve

BY D. Guy DATE 9/10/85 SUBJECT Water Tank Access and Vent Details SHEET NO ...... OF ......
CHKD. BY .... DATE .... JOB NO. ....

Scale 1" = 2'



RECEIVED
SEP 19 1985
UTAH STATE DIV. OF HEALTH
ENVIRONMENTAL HEALTH

APPROVED
UTAH STATE DIV. OF HEALTH
SAFE DRINKING WATER COMMITTEE
OCT 21 1985
REVIEW ENGINEER ________
EXECUTIVE SECRETARY ________

Figure 5B-4 Water Tank Access and Vent Details

Appendix 5-C

Subsidence Control and Monitoring

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Sections

Figures

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Attachments

SUBSIDENCE CONTROL AND MONITORING PLAN

SUBSIDENCE

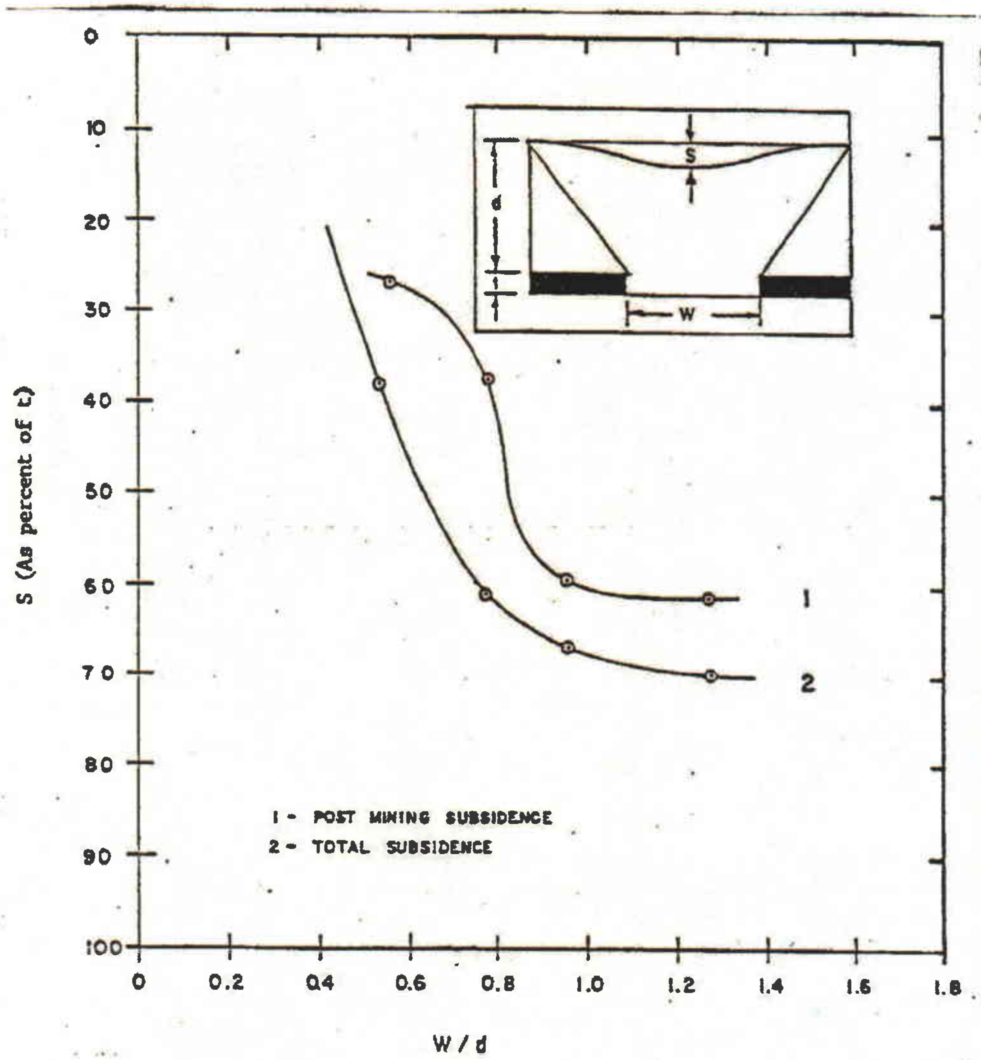
Subsidence can normally be expected to occur over areas where second mining has taken place ~~(pillaring)~~. See R645-301-523 for mining operation. Based on the geologic interruptions within a mine, subsidence becomes very difficult to predict, due to the variable nature of the mining panels. However, Figure 5C-1 will give an estimate of the maximum subsidence from room and pillar mining that may be expected in mine studied in the Western U.S. Maximum subsidence for an average room and pillar panel in the Bear Canyon Mine has been estimated from Figure 5C-1, using the criteria shown in Table 5C-1. For longwall panels, due to their ability to uniformly remove the coal, subsidence predictions are more accurate and there is less surface impacts. An analysis of subsidence effects from longwall mining specific to the Bear Canyon Mine reserves in the Tank seam and Hiawatha seam is included as Attachment 3. Attachment 3 mentions additional reserves, these reserves are located in the Blind Canyon. The cumulative affects of subsidence, based on Attachment 3 for the Tank and Hiawatha seams, and Attachment 1 for the Blind Canyon seam, is shown on Plate 5-3. Subsidence has been estimated based on the number of seams mined in the area and assuming the worst case scenario for mine layout and barrier pillar sizing.

For all subsidence calculations, and in determining the affected area an angle of draw of 22.5° was used. Past experience in this area shows no indication that subsidence would be this drastic, historically mines in the area have experienced an angle of draw of approximately 15°. Based on 18 years of subsidence evaluation at the Deer Creek Mine it was determined that 15° was an acceptable angle of draw. Their evaluations showed that only one area reported an angle of draw varying from <0° to 28° all other areas varied from <0° to 22°, with the <0° to 15° range being quite common. Due to the fact that the Bear Canyon mines are located across the canyon from the Deer Creek mines, and that all of the variables that exist with in the Deer Creek mines exist in the Bear Canyon mines similar results are expected. The maximum angle of draw of 25° to 30° quoted in Attachment 3 is the maximum angle of draw reported by the USBM study. for two seam extraction, between faulted areas, and is noted as being higher than average. Based on existing data from the Deer Creek mines and the Bear Canyon mines the 22.5° being used will project subsidence outside of the actual affected area, and is acceptable.

Additionally no actual subsidence has been noted from areas pillared by room and pillar method in the Bear Canyon #1 Mine as much as 40 years ago, and the subsidence monitoring network initiated in 1987, has shown only minor (0.47 ft max 1992) variations in elevation. Based on this, little, if any, detectable subsidence is expected to become apparent when mining under these depths. Some minor fracturing and an escarpment rock fall have been noted in the adjacent Trail Canyon Mine area, and although these are assumed to be mine-related, they occurred in areas of relatively low cover and unknown outcrop protection. Only minor fracturing has been noted in relation to the Bear Canyon Mine (see Plate 5-3). Based on this and on the environmental friendly design and mining methods being used, few surface fractures are anticipated. The main affect will be a uniform lowering in elevation.

Attachment 2 contains an evaluation of the escarpment stability for the Bear Canyon Mine Area. This report divided the Castlegate sandstone into 158 cells approximately 200 ft. wide. Full extraction mining has taken place under cells 104-111 and no escarpment failure was noticed. These cells were rated as stable in the report. Additionally full extraction mining has taken place under cells 21-34 and only one escarpment failure was noticed. This came from cell 32 which is rated as having a high instability factor. C. W. Mining personal only noticed one large rock fall accompanied by smaller gravel sized rocks and dust, once the initial rock hit the slope below the escarpment. All raptor nests are located in cells rated as stable except nest 920 inside cell 4 which has an instability rating of medium. Based on this, few impacts are expected to raptor nests. However a mitigation plan will be developed for all nest located inside the potential subsidence area as described on page 3-68.

Figure 5C-1 Subsidence Chart



Maximum subsidence, as a percentage of seam thickness, versus width/depth ratio for room and pillar mining at Somerset, Colorado (after Dunrud, 1980).

Table 5C-1 Estimated Maximum Subsidence

Coal Seam	Fee and Fed Lease U-024318	Federal Lease U-024316 and U-46484	Federal Lease U-020668 and U-38727	Fee and Federal Lease U-61049	Federal Lease U-61048
BLIND CANYON SEAM					
Panel Width	600 ft.	Mining Questionable	650 ft.	Not Minable	Not Minable
Average Depth	800 ft.		1200 ft		
Width/Depth Ratio	0.75		0.54		
Seam Thickness	9 ft.		9 ft.		
Maximum Calculated Subsidence	5.4 ft.		3.2 ft.		
HIAWATHA SEAM					
Panel Width	600 ft.	Mining Questionable	Not Minable	650 ft.	650 ft.
Average Depth	860 ft.			1600 ft.	1600 ft.
Width/Depth Ratio	0.75			0.40	0.40
Seam Thickness	5 ft.			14 ft.	14 ft.
Maximum Calculated Subsidence	3.2 ft.			5 ft.	5 ft.
TANK SEAM					
Panel Width	650 ft.	650 ft.	650 ~~650~~ ft.	650 ft.	Not Minable
Average Depth	560 ft.	1,400 ft.	1,400 ~~950~~ ft.	1,400 ft.	
Width/Depth Ratio	1.16	0.46	0.46 ~~0.68~~	0.17	
Seam Thickness	8 ft.	7 ft.	7.5 ft	7.5ft.	
Maximum Calculated Subsidence	5.5 ft.	4.5~~1.9~~ ft.	4.5~~4.1~~ ft.	4.5 ft.	
Total Calculated Subsidence	**14.1 ft.**	**4.5~~1.9~~ ft.**	**7.7 ~~7.3~~ ft.**	**10 ft.**	**5 ft**

MONITORING

Since subsidence may occur over any underground extraction, a monitoring network was installed in the summer of 1987, and has been monitored since that time. Monitoring stations are steel rebar with aluminum caps, set in concrete so weather, frost heave or wildstock/wildlife will not disturb them. Location of monitoring stations are shown on Plate 5-3.

Ten permanent subsidence monitoring points (SMS-1 thru SMS-5, Con 6, and SMS-7 thru SMS-10) are located on the mine site area. Before expansion into the Federal Lease area U-023416 the subsidence monitoring plan consisted of 3 monitoring points (SMS-1, SMS-2, and SMS-3) in the Bear Canyon Permit Area, a fourth point SMS-4 in the Trail Canyon Permit Area, and a Control Point CON-5, located outside the mining area. SMS-1, SMS-3, and SMS-4 are common to both the Trail Canyon and Bear Canyon Permits. CON-6 and SMS-7 thru SMS-10 were proposed in 1990 and established 22 September 1991. CON-5 became an additional subsidence monitoring point (redesignated SMS-5). The location of all existing and proposed points are shown on Plate 5-3.

15 additional monitoring points were installed on Federal Lease U-024316. These are shown on Plate 5-3 as points 11 through 24. 26 ~~additional~~ points ~~will be~~ were added to Wild Horse Ridge (Nos. 25 through 50) to monitor subsidence on Federal Lease U-020668 and U-38727. 7 points were added in and around Lease U-61049 (Numbers 51-57). Above the first long-wall panel 11 points were established at a spacing of 50 ft

going across the width of the panel (points 25A-K) in order to determine if subsidence is following the predicted pattern. Potential points were also selected above the 2nd and 3rd longwall panels and will continue to be selected above each of the panels one year prior to mining at a spacing of 250 ft as recommended in Attachment 3. The actual spacing and location of these points may change based on the results from points 24A-K, and on the yearly analysis that will be performed. If subsidence occurs as anticipated the spacing of 250 ft will continue. If it does not additional points will be added to determine the behavior, and our subsidence model will be updated. At a minimum 1 point will be placed in each panel as near as possible to the latitudinal and longitudinal centers.

Stations shall be monitored, and evaluated yearly for changes in elevation. This evaluation will include the current year and the previous two years at a minimum. In addition, a field investigation shall be made yearly of the mining area (including escarpment areas), and any obvious subsidence or mine related surface effects will be noted and located on a map. A copy of the results of the subsidence analysis, survey and map will be available for inspection at the office, and a summary of the ~~survey~~ results will be sent to the Division with the Annual Report.

MITIGATION/PROTECTION OF POTENTIAL IMPACTS

~~.11.20025Potential impacts and mitigation efforts are discussed in R645-301-5~~ Mr. Larry Dalton, Resource Analyst Utah Division of Wildlife Resources and the State's foremost authority on potential impacts of subsidence on wildlife, inspected the site in June 1984. The results of that investigation, as well as others, in part are as follows:

~~Considering the absence of spring, water sources, the negative potential impacts of subsidence within the Bear Canyon Permit Area could easily be offset by potential positive aspects.~~

On the negative side: Loss of riparian area and/or water sources and state appropriated water supply~~rights~~ is of greatest concern, followed by loss of vegetation from methane gas leaking to the surface from an underground works. Considering the lack of riparian area or water sources above the coal seam, this concern is not warranted for most areas. There are two area of concern above Fish Creek in section 19 as shown on Plate 7-4. These areas will be monitored for loss of water as it is being undermined. ~~Secondly~~, In regards to methane gas Co-Op has never encountered methane gas underground so there is little concern relative to potential vegetation loss. ~~, and last,~~ The last concern is the loss of nests due to escarpment failure.

On the positive side: The tension fractures resulting from subsidence along the steep side hills are frequently utilized by big game as movement corridors. The fractures and rubble provide escape cover for a variety of wildlife species as well as additional habitat for burrowing and denning animals. While there is concern over the potential loss of nests as a result of escarpment failure, there is also a potential for additional nesting sites to be created through this gravitational shearing of escarpment surfaces.

PROTECTION

In order to protect water resources and state appropriated water supplies from impacts C. W. Mining has designed their mine layout so that areas where these resources exist with less then 900 feet of overburden between the resource and the coal, the resource will be outside of the affected area. Based on the mining handbook[1] and past history, 900 feet of overburden is sufficient to prevent adverse affects to the resource. ([1] Lowrie, Raymond L., ed. 2002 "SME Mining Reference Handbook" pp. 256)

Additionally in the areas where perennial streams exist above the affected area C. W. Mining will increase the monitoring of these areas to a weekly basis one month prior to mining in the area. This weekly monitoring will continue until one month after mining has left the area. Monitoring will then be reduced to once a month for an additional 6 months at which time it will resume its normal schedule. This increased monitoring will include the sites FC-2, FC-3, FC-4, FC-5, and SCC-2 for the right fork of Fish Creek, and FC-1, FC-6, SBC-18, SBC-16, SBC-16A, SBC-16B, SBC-20, and SBC-21 for the left fork of Fish Creek.

This monitoring will take place as each panel passes under the area. During the monitoring weekly reports will be sent to the Division via email.

In potential escarpment failure areas containing raptor nests C. W. Mining will develop a mitigation plan by July 1, 2007 as described on page 3-68. The plan will be included in Appendix 3L once it is complete.

MITIGATION~~NOTIFICATION~~

Notification

During operation, all owners of property within the area that could be impacted by subsidence shall be notified by mail six months prior to mining beneath their property and be informed of:

a. Specific areas mining will take place

b. Dates of underground operations that could cause subsidence in the area.

c. Measures to be taken to prevent and or control adverse surface effects.

Co-Op further commits to the following course of action should subsidence cause any material damage or a reduction in value of structure or land.

a. Restore, rehabilitate, or remove and replace, to the extent technologically and economically feasible, each materially damaged structure, feature or value promptly after the material damage from subsidence is suffered, to the condition it would be in if no subsidence had occurred and restore, to the extent technologically and economically feasible, those surface lands that were reduced in reasonable foreseeable use as a result of such subsidence to a condition capable of supporting before subsidence; or

b. Purchase the damaged structure or feature (except structures or features owned by the person who conducted the underground coal mining activities) for its pre-subsidence fair market value. The person conducting the underground coal mining operation shall promptly, after the material damage or reduction in value or reasonable foreseeable use from subsidence occur, to the extent technologically and economically feasible, restore the purchased structure or the structure owned by the person conduction the underground mining operations, restore those surface lands that were materially damaged or reduced in value or reasonable foreseeable use by such subsidence, to a condition capable and appropriate of supporting the structure, and any other foreseeable uses such surface lands were capable of supporting before mining. Nothing in the paragraph shall be deemed to grant or authorize an exercise of the power of condemnation of the right of eminent domain by any person engaged in underground coal mining activities; or

c. Compensate the owner of any surface structure in the full amount of the diminution in value resulting from subsidence, by purchase prior to mining of a noncancellable premium prepaid insurance policy or other means approved by the Division as assuring before mining begins that payments will occur; identify every person owning an interest in the surface for all damages suffered as a result of the subsidence; and , to the extent technologically and economically feasible, fully restore the land to a condition capable of maintaining reasonably foreseeable uses which it could support before subsidence.

d. The area will be monitored on an annual basis, and field investigation will also be performed at that time. If escarpment failure is observed in areas where no escarpment failure is anticipated, ~~mining will be immediately stopped in the~~An immediate evaluation of the ~~e~~affected area, ~~until a proper evaluation can~~will be performed to determine the cause of the failure, and any necessary remedies or protection required. The DOGM and the U.S. Forest Service, Price District Ranger would be notified of such an occurrence. An escarpment stability study is included as attachment 2.

U-024316

Mining is projected within the vicinity of Bear Creek in Federal Lease U-024316. An additional concern over escarpment failure has been raised by the U.S. Forest Service; therefore, the following discussion will address the potential for such failure.

The steep area of Bear Canyon in the S.W. corner of Section 13 is approx 1200 feet above the coal seam (See Plate 6-10). To prevent subsidence to Bear Creek and the adjacent ledges, no retreat mining was conducted East of the in-mine fault paralleling the section line between sections 13 and 14, T.16S., R.7.E. See Plate 3-4C.

This Protection Zone was determined by examining the angles of draw. Figure 5C-2 shows angle of draws plotted against depths for various mines within the general area of the Bear Canyon Mine. Based on this information, an average angle of draw of 22.5° was used.

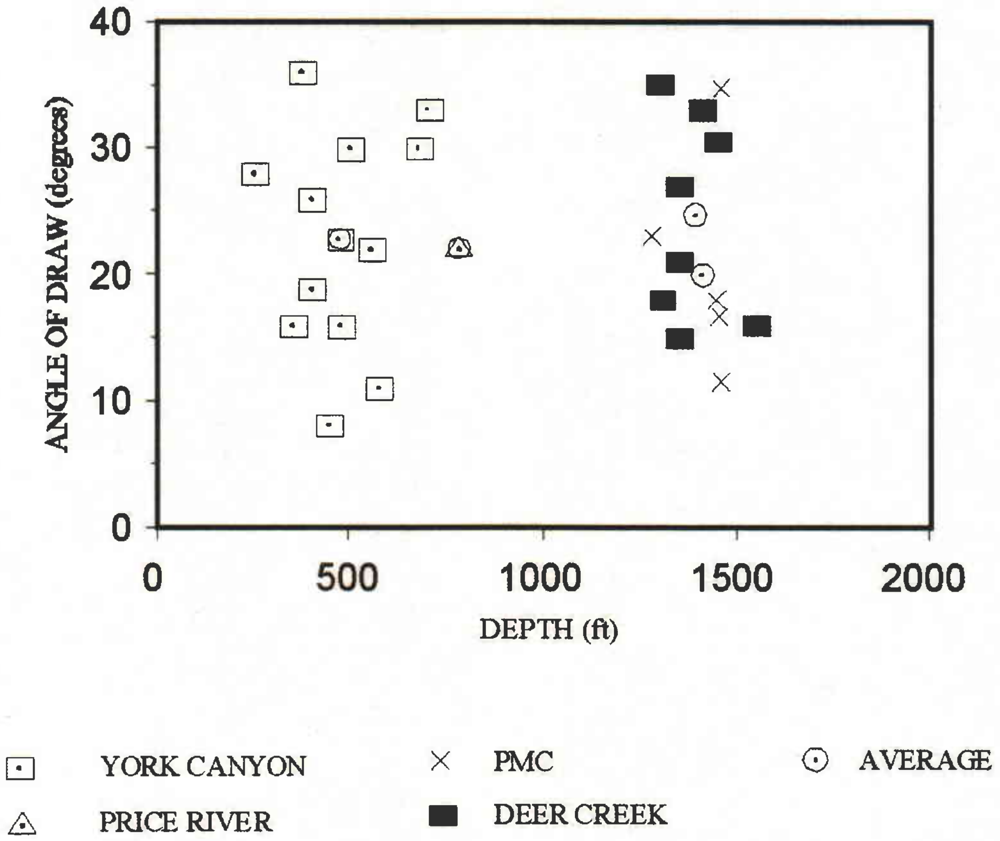
Plate 5-3 shows the areas which will potentially be affected by subsidence. None of the area within Federal Lease U-024316 is included in this potential area..

Within the lease, pillars were developed on 80' centers minimum. This pillar size was evaluated using the "Analysis of Retreat Mining Pillar Stability (ARMPS)" software, developed by the National Institute for Occupational Health and Safety. A pillar stability factor of 1.54 was determined, which shows the pillar size to be adequate to prevent subsidence of the ledges and Bear Creek. NIOSH research has found that in 94% of all case studies stability factors greater than 1.5 have provided long-term stability (Mark, 1997), showing that pillars with safety factors above 1.5 are adequate to hold the weight of the overburden, thus preventing subsidence. Monitoring of the ledges for subsidence subsequent to the development on this lease has confirmed that no subsidence has occurred, and no escarpment failures have been observed.

Outcrop protection has been increased to a minimum of 200 feet in this area ~~the plan (see R645-301-525-300).~~ This is consistent with other mines in the Wasatch Plateau, and with the exception of some longwall operations, has been shown to be effective at preventing escarpment failure near outcrops.

Figure 5C-2

Subsidence Factor Versus Mining Depth



U-38727, U-61048, U-61049, and U-020668

As with Federal Lease U-024316, lease stipulation 13~~12~~ requires mining to be conducted in a manner to prevent hazardous conditions such as potential escarpment failure.

The uppermost escarpment in the Wild Horse Ridge area is the Castlegate sandstone, located approximately 800 ft. above the Tank Seam, and 950 ft. Above the Blind Canyon Seam.

To prevent subsidence to these escarpments in areas where it has been determined escarpment failure would be a hazardous condition, a barrier zone will be left in which no retreat mining will take place. The width of this barrier was determined using an angle of draw of 22.5° (See Figure 5C-2 and attachment 2).

~~or the Tank Seam a minimum barrier of 300' will be maintained in which no retreat mining will take place. For the Blind Canyon Seam, a minimum barrier of 370 feet will be maintained.F~~ Plate 5-3 shows the cumulative anticipated ~~zone which will be affected by~~ subsidence contours, and the Castlegate Sandstone, located ~~outside of this area.~~within the permit area. ~~This zone is also~~Individual seam subsidence contours are shown on Plates 5-1A, 5-1B, and 5-1C to show the relationship between the development and retreat panels. W~~. Therefore, this pillar size will be adequate to prevent subsidence and escarpment failure. Mining on Federal Lease U-024316 has also confirmed that this size is adequate in the Bear Canyon Mine area. for room and pillar retreat mininghere panels are shown encroaching on the barrier zone, pillars will be developed and left in~~

~~place to prevent subsidence. The pillars will be developed on 80' centers minimum. Using the ARMPS software (NIOSH), a minimum pillar stability factor of 1.58 was determined~~

There area two areas within these leases and a third area just outside of the leases where it has been determined that escarpment failure does not present a hazardous condition. The locations of the areas are in the left fork of Fish Creek where it runs through lease U-020668, and U-38782, as well as an area at the top of the left fork of Fish Creek just outside of two portions of lease U-61049, and in the left fork of Bear Creek where it runs through lease U-61049. These areas as well as additional areas of been studied and modeled for rock falls. This study is included as Attachment 2. A summary and discussion of these result are included below. The cross-sections modeled for rock falls are shown on Plate 5-3.

Summary of Rock Fall Anaylsis

Section	Distance to Stream Bed	Maximum Rockfall Distance
A-A'	2,050 ft.	800 ft.
B-B'	1,674 ft.	1,200 ft.
C-C'	2,600 ft.	950 ft.
D-D'	1,980 ft.	650 ft.
E-E'	450 ft.	450 ft. (rock hits bottom of canyon)

Section A-A'

This area is located above the old Bear Canyon #1 and #2 mines. It was used to calibrate the model

Section B-B'

This section is located on the point of Wild Horse Ridge. It was initially selected because it represented the steepest slope within the affected area. However as mining proceeded towards this area it was discovered that there was active burning so mining stopped and never reached this area. This area will however be impacted by natural subsidence resulting from the natural burning of the coal.

Section C-C'

This section is located on Wild Horse Ridge against the left fork of Fish Creek near the south-east end of U-38727. It was selected because secondary mining will take place under this area and also go out past the escarpments. The escarpments in this area range form 0-80 feet. The cross-section was placed where escarpments were the largest and the slope was the steepest. Escarpment failure will occur in this section, however based on models, the failure will not reach the stream channel so no water impacts will occur. There will however be loss of vegetation in the path of the rock fall. This will have minimal aesthetic impacts since there is little vegetation along the slope and also because escarpment failure happens naturally along Fish Creek so any areas would still match the appearance of surrounding areas.

Section D-D'

This section is located on Wild Horse Ridge against the left fork of Fish Creek near the north-east end of U-38727. This section represents the transition area where subsidence contours are beginning to move from under the escarpments to adjacent to the escarpments, and then away from the escarpments. The escarpments in this area range from 80-160 feet. Any escarpment failure in this area will not reach the stream channel so impacts are the same as section C-C'.

Section E-E'

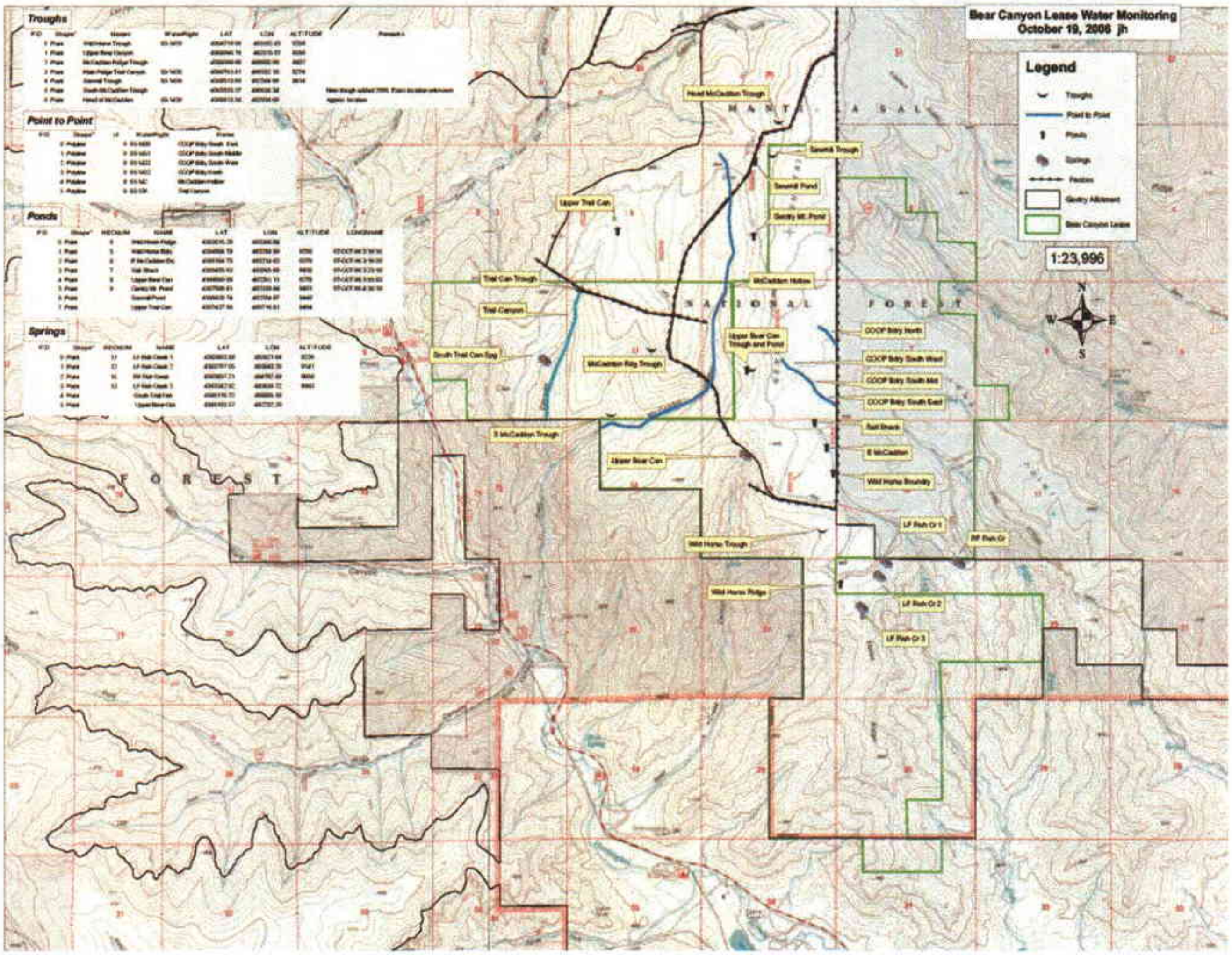
This area is at the upper portion of the right fork of Fish Creek between the two segments of Lease U-61049. Fish Creek is a box canyon and the escarpments in the area that will be impacted are the stream bed. The escarpments range from 160-240 feet. Since the escarpments are the stream bed any escarpment failure would have an impact on water resources. However the impacts would be quickly dissipated since flow are minimal in this area (10-30 gpm). Little vegetation impact is expected because of the short slope distance and the fact that water has eroded most of the soil in the area leaving exposed rock ledges.

Lease stipulation 21 requires the replacement of all water sources identified for protection. Figure 5C-3 shows all water sources identified for protection. In accordance with the lease stipulation all sources identified will be replaced if impacted. Several of the sources identified are within the subsidence area and have been selected as water monitoring points as shown on Plate 7-4. The relationship between the water monitoring points and the names identified on figure 5C-3 is outlined below.

F. S. Name	Monitoring Name	Location
LF Fish Cr 3	SBC-16	Inside subsidence zone
LF Fish Cr 2	SBC-16A	Inside subsidence zone
LF Fish Cr 1	SBC-16B	Inside subsidence zone
RF Fish Cr	SBC-18	Inside subsidence zone
Wild Horse Ridge	SBC-22	Inside subsidence zone
Wild Horse Boundary	none (Inspected with FC-6)	Outside subsidence zone
E McCadden	none (Inspected with FC-6)	Outside subsidence zone
Salt Shack	none (Inspected with FC-6)	Outside subsidence zone
S McCadden Trough	SMH-3	Outside subsidence zone
McCadden Rdg Trough	SMH-2	Inside subsidence zone
Upper Bear Can Trough	SMH-5	Outside subsidence zone
COOP Bdry South East	none (Inspected with FC-7)	Outside subsidence zone
COOP Bdry South Mid	none (Inspected with FC-7)	Outside subsidence zone

COOP Bdry South West	none (Inspected with FC-7)	Outside subsidence zone
COOP Bdry North	none (Inspected with FC-8)	Inside subsidence zone
McCadden Hollow	none (Inspected with SMH-4)	Inside subsidence zone
Gentry Mt Pond	none	Outside permit area
Sawmill Pond	none	Outside permit area
Sawmill Trough	none	Outside permit area
Head McCadden Trough	none	Outside permit area
Upper Trail Can	none	Outside permit area
Trail Can Trough	FBC-1	Outside subsidence zone
Trail Canyon	none	Outside subsidence zone
South Trail Can Spg	FBC-8	Outside subsidence zone

Figure 5C-3 Forest Service Protected Water Resources



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Mark, C., and F.E. Chase
Analysis of Retreat Mining Pillar Stability (ARMPS). Paper in Proceedings on New Technology for Ground Control In Retreat Mining, 1997, NIOSH pub. 97-133, pp. 17-34.

ANALYSIS OF RETREAT MINING PILLAR STABILITY (ARMPS)

By Christopher Mark, Ph.D.,[1] and Frank E. Chase[2]

ABSTRACT

The prevention of pillar squeezes, massive pillar collapses, and bumps is critical to safe pillar recovery operations. To help prevent these underground safety problems, the Pittsburgh Research Center has developed the Analysis of Retreat Mining Pillar Stability (ARMPS) computer program. ARMPS calculates stability factors (SF) based on estimates of the loads applied to, and the load-bearing capacities of, pillars during retreat mining. The program can model the significant features of most retreat mining layouts, including angled crosscuts, varied spacings between entries, barrier pillars between the active section and old (side) gobs, and slab cuts in the barriers on retreat. It also features a pillar strength formula that considers the greater strength of rectangular pillars. The program may be used to evaluate bleeder designs, as well as active workings.

A data base of 140 pillar retreat case histories has been collected across the United States to verify the program. It was found that satisfactory conditions were very rare when the ARMPS SF was less than 0.75. Conversely, very few unsatisfactory designs were found where the ARMPS SF was greater than 1.5. Preliminary analyses also indicate that pillar failures are more likely beneath sandstone roof and that the ARMPS SF may be less meaningful when the depth of cover exceeds 230 m (750 ft).

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INTRODUCTION

The use of remote-control continuous miners, extended cuts, and mobile roof supports has increased the productivity of room-and-pillar retreat mining (also referred to as "pillaring," "pillar recovery," "robbing," and "second mining"). In the southern Appalachian coalfields, many mines are choosing room-and-pillar retreat mining because of its lower capital cost and greater flexibility [Blaiklock 1992]. Unfortunately, between 1989 and 1996, 25% of all roof and rib fatalities occurred on pillar recovery sections.

Roof fall accidents are not the only problem associated with retreat mining. Millions of tons of coal are sterilized annually because of pillar squeezes, floor heave, pillar line roof falls, and pillar bumps. Traditional pillar design methods are of little help due to the complex mining geometries and abutment pressures that are present during pillar extraction. The Pittsburgh Research Center has developed the Analysis of Retreat Mining Pillar Stability (ARMPS) computer program to aid in the design of pillar recovery operations. This paper describes the program and presents the findings thus far.

THE ARMPS METHOD

The goal of ARMPS is to help ensure that the pillars developed for future extraction (production pillars) are of adequate size for all anticipated loading conditions. The key is to be able to estimate the magnitudes of the various loads that the pillars might experience throughout the mining process. The formulas used in ARMPS are based on those originally developed for the Analysis of Longwall Pillar Stability (ALPS) method, which is widely used for longwall pillar design [Mark et al. 1992]. ALPS was initially derived from underground measurements of longwall abutment stresses and was later validated by the back-analysis of more than 100 case histories. In ARMPS, the formulas have been extensively modified for the variety of mining geometries typically found in pillar recovery operations.

USER INPUT

The first step in using the ARMPS program is to enter the dimensions of the pillars in the working section, as illustrated in figure 1. The program can accommodate angled crosscuts, varied spacings between the entries, and barrier pillars between the active section and old (side) gob areas. Slabbing of barriers

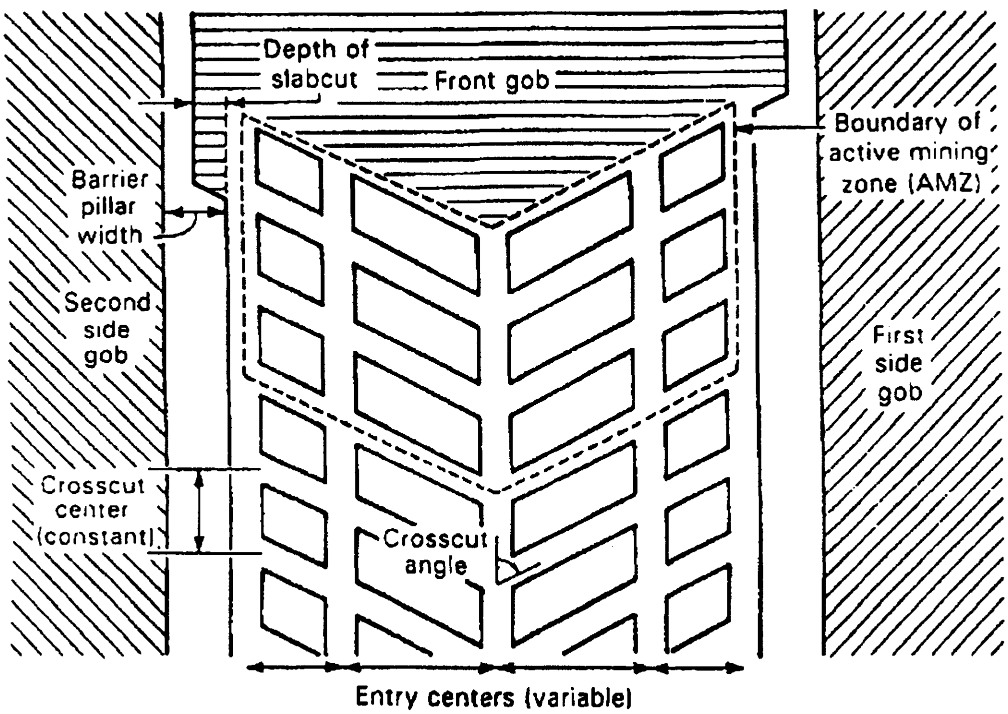


Figure 1.—Section layout parameters used in ARMPS.

on retreat can also be included. Other parameters that must be defined include depth of cover, mining height, entry width, and crosscut spacing. Finally, the user chooses one of four possible *loading conditions* (figure 2). The simplest, loading Condition 1, is development loading only. Loading condition 2 occurs when the active, or "front," panel is being fully retreated and there are no adjacent mined-out areas. The total applied load is the sum of the development loads and the front abutment load. Loading condition 3 occurs where the active mining zone (AMZ) is adjacent to an old (side) gob and the pillars are subjected to development, side abutment, and front abutment loads. Where the pillar line is surrounded by gob on three sides (sometimes referred to as "bottlenecking"), loading condition 4 is used. In every case, the extent of each gob is defined by the user.

ARMPS STABILITY FACTOR FOR THE ACTIVE MINING ZONE

The basic output from the ARMPS program is the stability factor (SF), defined as

$$\text{ARMPS SF} = \text{LBC/LT}, \qquad (1)$$

where LBC = the estimated total load-bearing capacity of the pillars within the AMZ,

and LT = the estimated total load applied to pillars within the AMZ.

Figure 3 illustrates the development and front abutment loads applied to the AMZ.

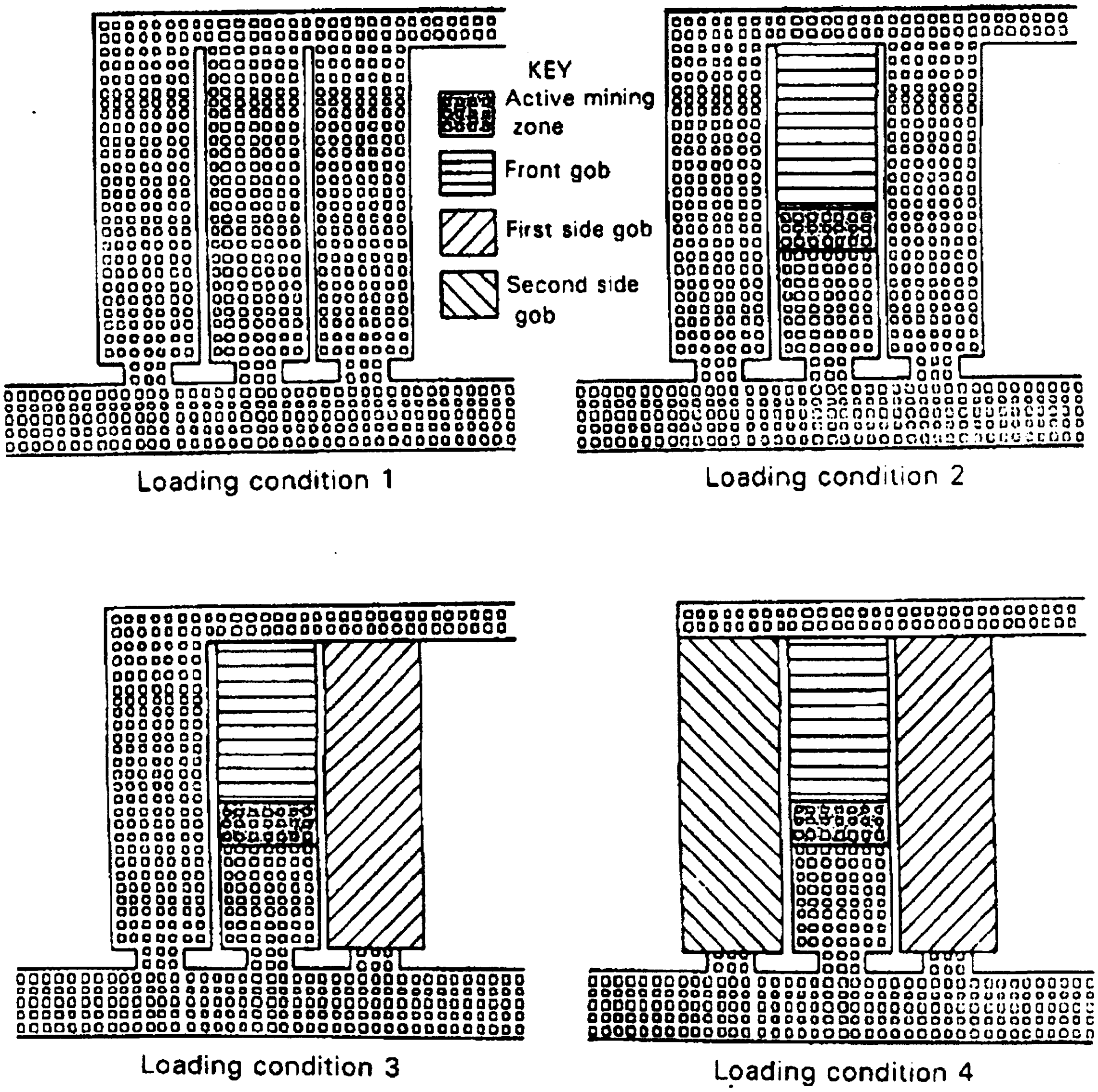
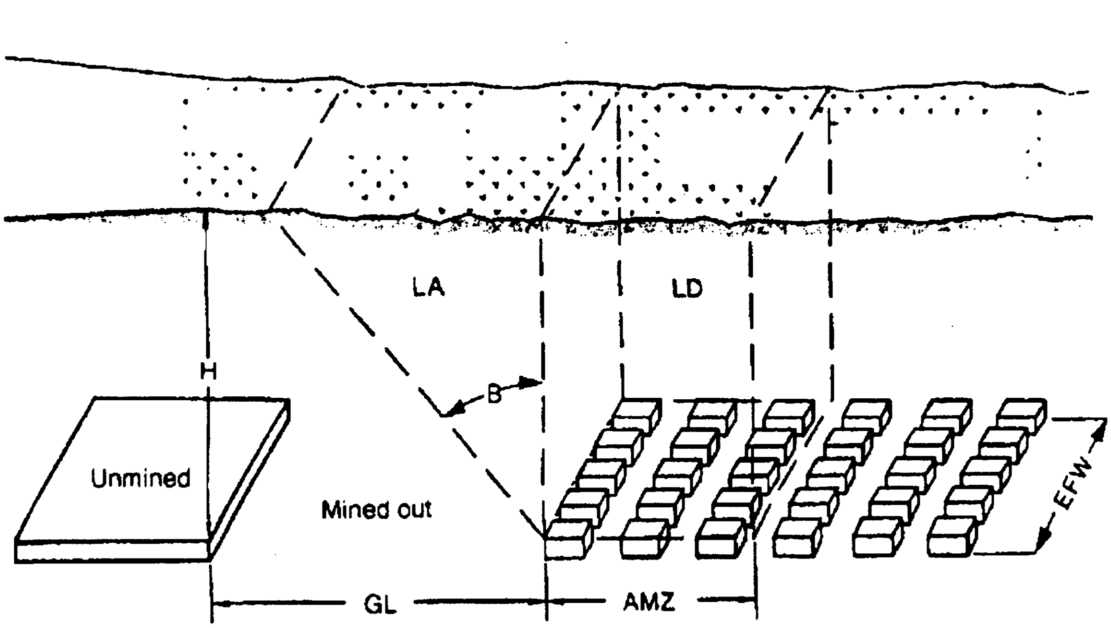


Figure 2.—The four loading conditions that can be evaluated with ARMPS.



KEY

AMZ Active mining zone
B Abutment angle
EFW Extraction front width
GL Mined out area
H Depth of cover
LA Abutment load
LD Development load

Figure 3.—Schematic showing the active mining zone, the development load, and the front abutment load.

The AMZ includes all of the pillars on the extraction front (or "pillar line") and extends outby the pillar line a distance of five times the square root of the depth of cover ($5\sqrt{H}$). This distance was selected because measurements of abutment stress distributions [Mark 1990] show that 90% of the front abutment load falls within its boundaries (figure 4).

ARMPS calculates the SF for the entire AMZ, rather than stability factors for individual pillars, because experience has shown that the pillars within the AMZ typically behave as a *system*. If an individual pillar is overloaded, it will normally transfer its excess load to adjacent pillars. If those pillars are adequately sized, the process ends there. A pillar squeeze occurs only when the adjacent pillars are also undersized. They then fail in turn, resulting in a "domino" of load transfer and pillar failure. The ARMPS SF is therefore a measure of the overall stability of the pillar system.

PILLAR LOAD-BEARING CAPACITY

The load-bearing capacity of the AMZ is calculated by summing the load-bearing capacities of all of the pillars within its boundaries. The strength of an individual pillar (SP) is determined using a new pillar strength formula (the Mark-Bieniawski formula) that considers the effect of pillar length:

$$SP = S_1 [0.64 + (0.54 - 0.18 (w^2/hL))], \qquad (2)$$

where S_1 = in situ coal strength, assumed = 6.2 MPa (900 psi),

w = pillar width,

h = pillar height,

and L = pillar length.

The new pillar strength formula was needed because the pillars used in retreat mining are often much longer than they are wide. The strength of rectangular pillars can be significantly greater than square pillars due to the greater confinement generated within them. The Mark-Bieniawski formula was derived from analyses of the pillar stress distributions implied by empirical pillar strength formulas. A complete discussion of the Mark-Bieniawski formula is included in appendix A of this paper. The in situ coal strength is assumed to be 6.2 MPa (900 psi) in ARMPS; however, this value can be modified by the user.

The load-bearing capacity of the pillars is determined by multiplying their strength by their load-bearing area. When angled crosscuts are employed, the algorithm still calculates accurately each pillar's least dimension, length, and load bearing area (A_p):

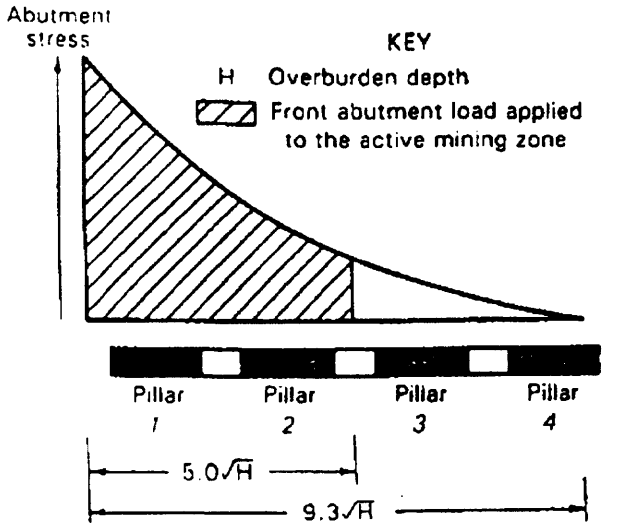


Figure 4.—Distribution of abutment stress, showing that 90% of the abutment falls within the distance of (5√H) from the gob edge.

$$A_p = [(XC)(ECTR) - (XC)(W_e) - (ECTR)(W_e)/(\sin\phi) + (W_e)^2/(\sin\phi)], \tag{3}$$

where XC = center-to-center crosscut spacing,

$ECTR$ = center-to-center entry spacing,

W_e = entry width,

and ϕ = angle between the crosscut and the entry.

The load-bearing capacity of the pillar system is then obtained by summing the capacities of the individual pillars within the AMZ. ARMPS calculates the strength and load-bearing capacity of barrier pillars in the same manner as the panel pillars, except that their length is limited to the breadth of the AMZ.

PILLAR LOADINGS

The loadings applied to the AMZ include development loads, abutment loads, and loads transferred from barrier pillars. Table 1 shows the sources of loads and the loading conditions in which they occur.

Table 1.—Loads applied to the active mining zone in ARMPS

Source of load	Loading condition 1	2	3	4
Development	X	X	X	X
Front abutment		X	X	X
Side gob abutments			X	X
Transfer from barriers between active mining zone and side gobs			X	X
Transfer from remnant barriers between front gob and side gobs			X	X

Development loads are due to the weight of the overburden directly above the pillars before any retreat mining takes place. The tributary area theory is used in ARMPS to estimate development loads.

Abutment loads occur as a result of retreat mining and gob formation. They are determined by the depth of cover, the extent of the gobs, the width of the extraction front, and the abutment angles. These parameters are illustrated in two dimensions in figure 5. The abutment angle determines how much load is carried by gob. Measurements of longwall abutment stresses indicated that an abutment angle of 21° is appropriate for normal caving conditions [Mark 1992]. The ARMPS program initializes the abutment angles for all gobs to 21°, however, this can be changed by the user. For example, if it is known that no caving has occurred, then the abutment angle may be set to 90° to simulate zero load transfer to the gob [Chase and Mark 1993].

The abutment stresses are assumed to be distributed following the inverse-square function shown in figure 4. Abutment loads are also applied to barrier pillars; however, if a barrier is too small to carry its share, then some or all of the excess is transferred to the AMZ.

The front abutment load applied to the AMZ is calculated as follows. The volume of the overburden above the mined-out active gob is the depth of cover multiplied by the gob area. The portion of this volume whose weight is carried by the gob is determined by the tangent of the abutment angle, as shown in figure 5. This portion is subtracted, and the remainder is shared between the AMZ and the unmined coal on the other three sides of the gob. It is assumed that barrier pillars (or substantial production pillars) are present on the other three sides of the gob. Load applied to the barriers here may be transferred back to the AMZ if the barriers are removed later in the mining process.

The magnitude of the front abutment load applied to the AMZ is determined by the extent of the extraction zone and the depth of cover. The front abutment is considered fully developed if the gob area is large relative to the depth of cover (figure 6A). If only a few rows of pillars have been extracted (figure 6B), much of the load will be carried by the back barrier. If the full extraction zone is rather narrow (figure 6C), much of the load will be carried by the side barriers.

The side abutment loads are shared by the AMZ and, if it is present, the barrier pillar between the AMZ and the side gob. The inverse-square stress distribution (figure 4) again is used to apportion the load between the barrier and the AMZ. Next, if it is determined that the barriers are overloaded, some additional side abutment load is transferred to the AMZ.

To determine whether a barrier pillar can carry the load applied to it, ARMPS estimates the barrier's SF by dividing its load-bearing capacity by its load. The total load applied to a barrier pillar is the sum of the development load, the front abutment load due to any slabbing, and the side abutment load applied to the barrier. If the SF is greater than 1.5, the barrier is assumed to be stable. When the barrier's SF is between 1.5

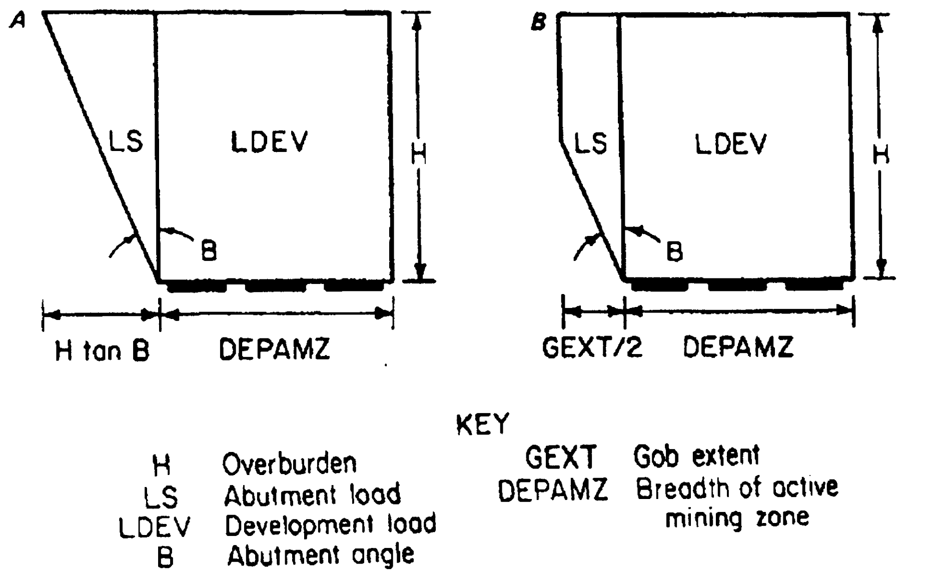


Figure 5.—Schematic showing the abutment load in two dimensions. *A*, supercritical gob; *B*, subcritical gob.

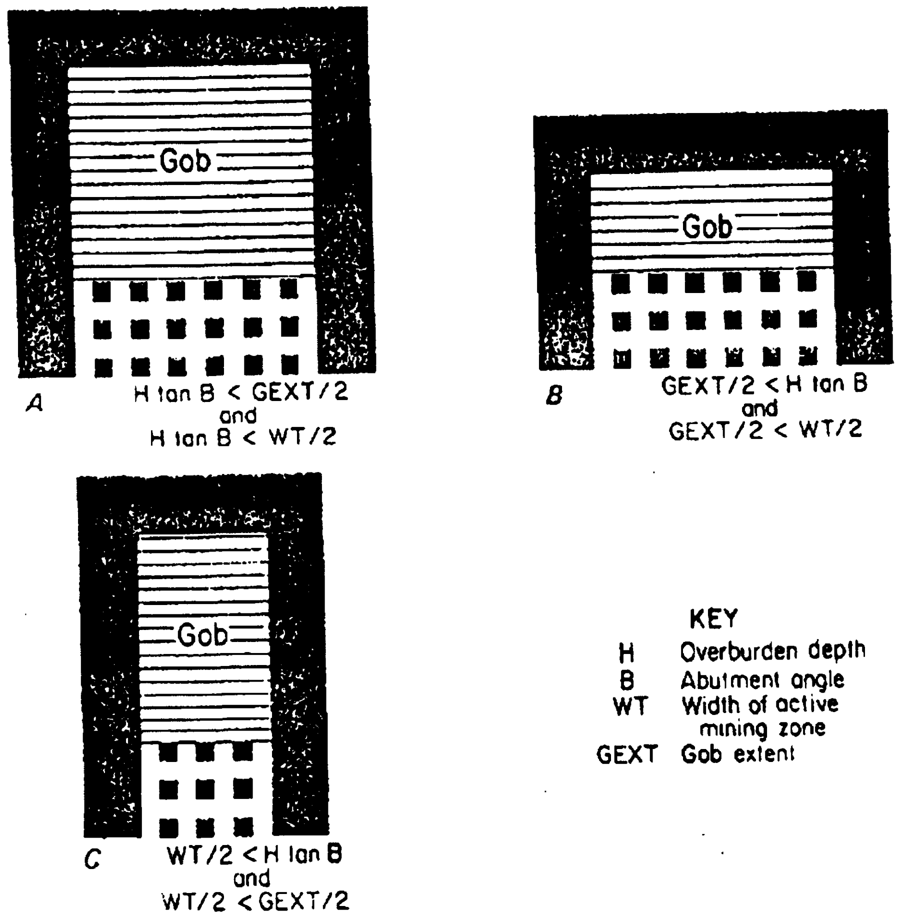


Figure 6.—Illustration of the effect of panel geometry on the front abutment loading in ARMPS. *A*, gob area is supercritical in both width and extent; *B*, gob area is subcritical in extent; *C*, gob area is subcritical in width.

and 0.5, a portion of its abutment load is transferred to the AMZ. If the SF is less than 0.5, all of the additional side abutment load (but not the development or front abutment load) is transferred to the AMZ.

The final sources of load on the AMZ are the remnant barrier pillars inby the pillar line (between the front and side gobs). If the remnant barriers are too small to carry their load, some part of it is returned to the AMZ. The decision to transfer the load and how much is based on the remnant barrier's SF. Slabbing of the remnant will also return some abutment load to the AMZ.

Further details on the formulas and calculations used in ARMPS loadings can be found in the "Help" text that accompanies version 4.0 of the program.

VERIFICATION OF THE ARMPS METHOD

The ARMPS method is being verified through back-analysis of pillar recovery case histories. To date, 140 case histories have been obtained from 10 States (see appendix B of this paper). They cover an extensive range of geologic conditions, roof rock cavability characteristics, extraction methods, depths of cover, and pillar geometries. Ground conditions in each case history have been categorized as either satisfactory or unsatisfactory. Pillar failures responsible for unsatisfactory conditions were found to include—

- Pillar squeezes, accompanied by significant entry closure and loss of reserves;
- Sudden collapses of groups of pillars, usually accompanied by airblasts; and/or
- Coal pillar bumps (violent failures of one or more pillars).

As figure 7 shows, pillar failures occurred in 93% of the cases where the ARMPS SF was less than 0.75. Where the ARMPS SF was greater than 1.5, 94% of the designs were satisfactory. SF values ranging from 0.75 to 1.50 form a "gray" area where both successful and unsuccessful cases are found.

Current research has begun to evaluate other factors that may contribute to satisfactory conditions when the ARMPS SF falls between 0.75 and 1.5. These include—

Coal strength: An extensive data base of laboratory tests of the strength of coal was compiled by Mark and Barton [1997]. When compared with the ARMPS data base, no correlation was found between coal strength and pillar strength.

Depth of cover: Figure 8 shows that there is a marked reduction in SF as depth of cover increases. When the depth exceeds 305 m (1,000 ft), the ARMPS SF was below 1.0 for 70% of the satisfactory designs. Highly unsatisfactory conditions have also been encountered under deep cover, which recently led to two fatalities. Pillar design for retreat mining under deep cover remains an important research issue.

Seam height: A plot of seam height against ARMPS SF shows no correlation (figure 9).

Roof geology: A detailed study of pillar performance was conducted at a mining complex in southern West Virginia. More than 50 case histories were collected. Analysis showed that satisfactory conditions were more likely to be encountered under shale roof than massive sandstone roof (figures 10-11). This implies that better caving occurs with shale, resulting in lower pillar loads.

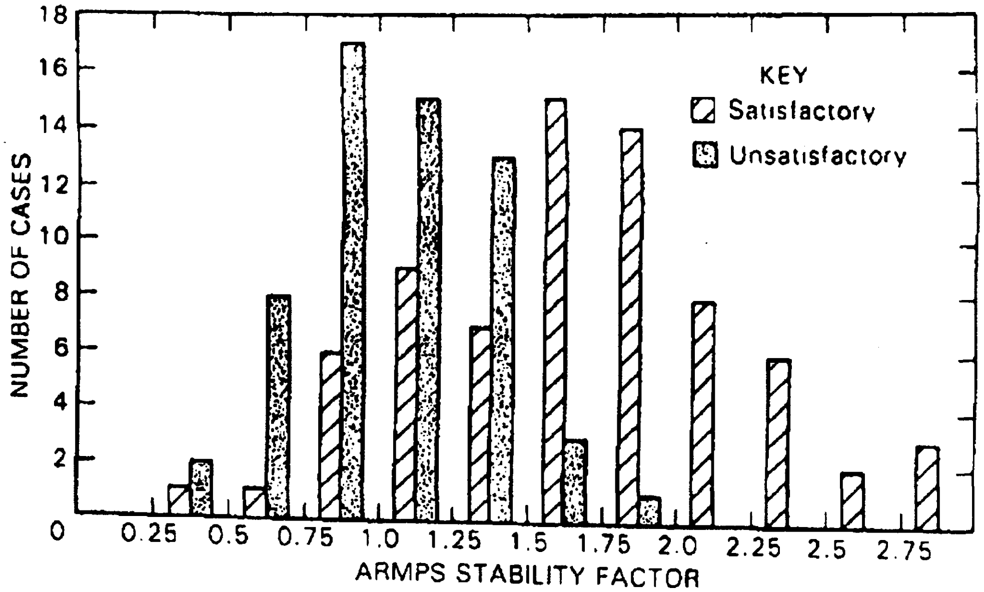


Figure 7.—ARMPS data base.

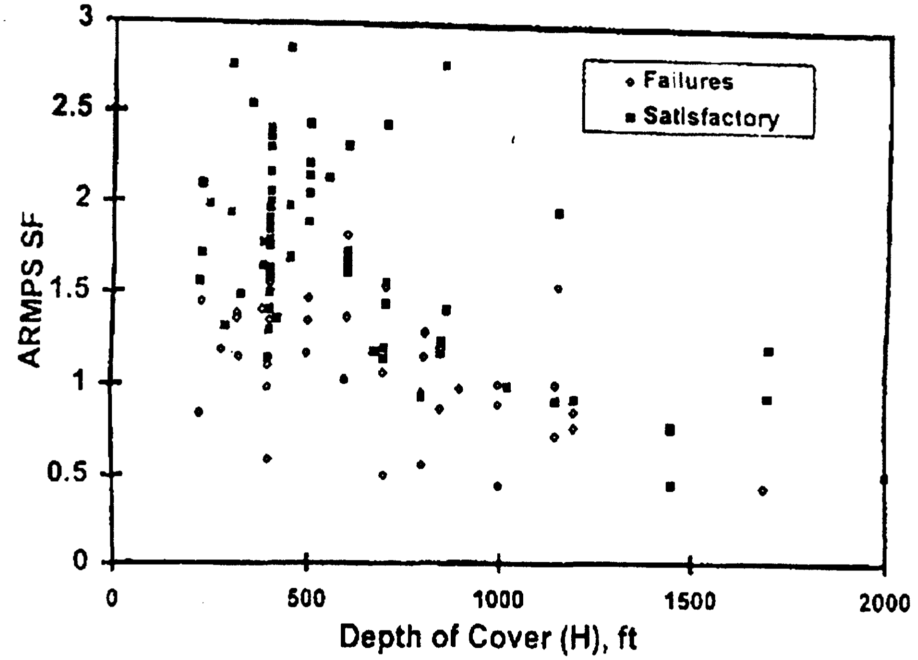


Figure 8.—Relationship between ARMPS SF and depth of cover within the case history data base.

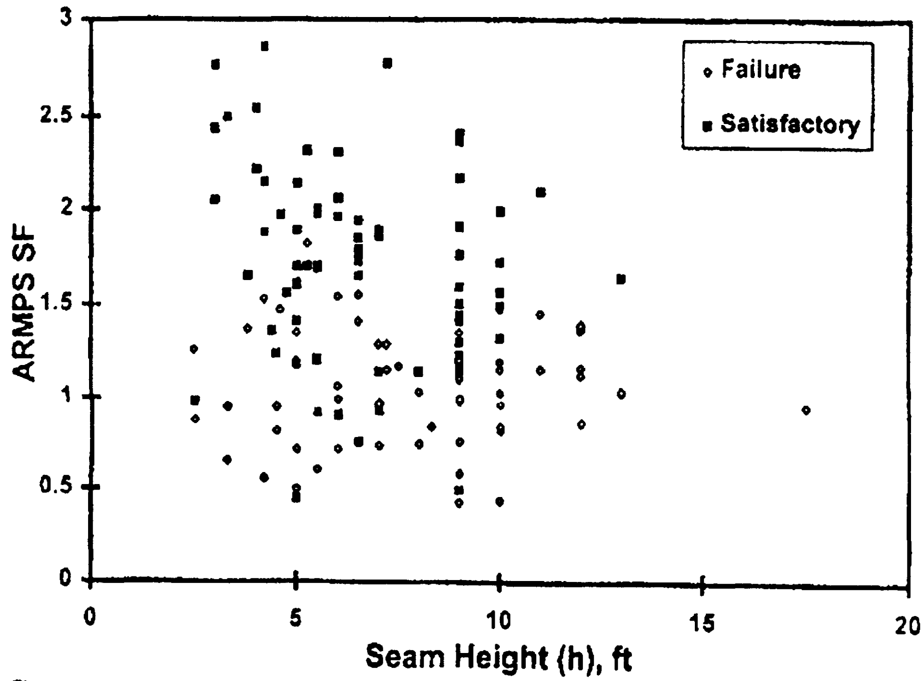


Figure 9.—Relationship between ARMPS SF and seam height within the case history data base.

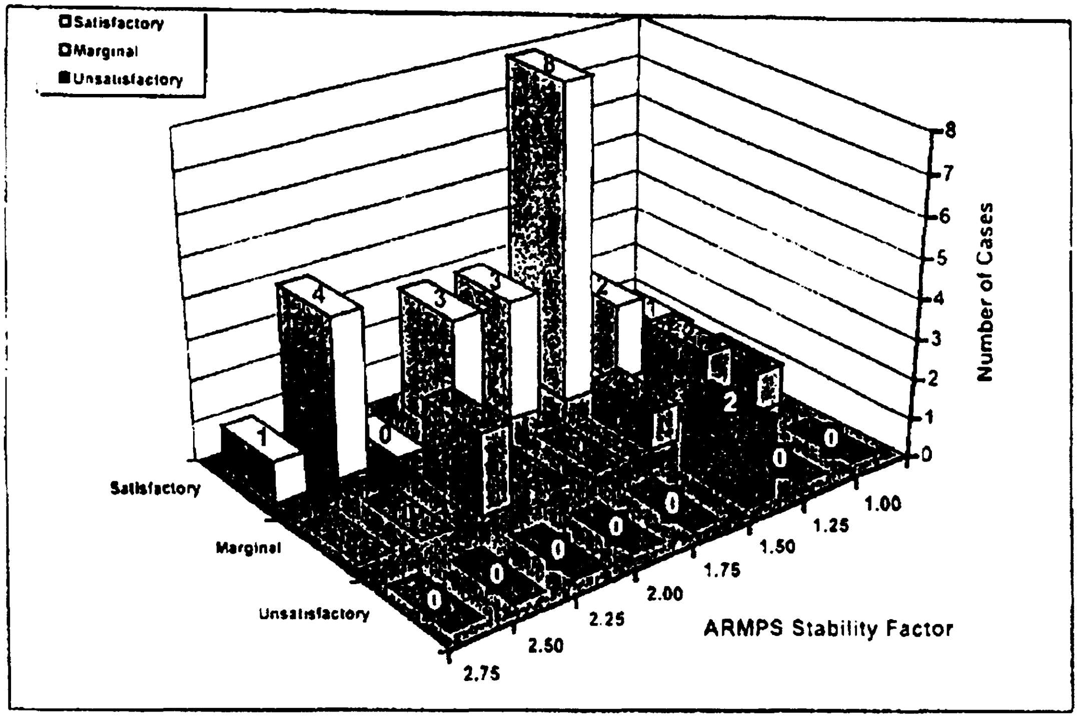


Figure 10.—Shale roof case histories from mining complex in southern West Virginia.

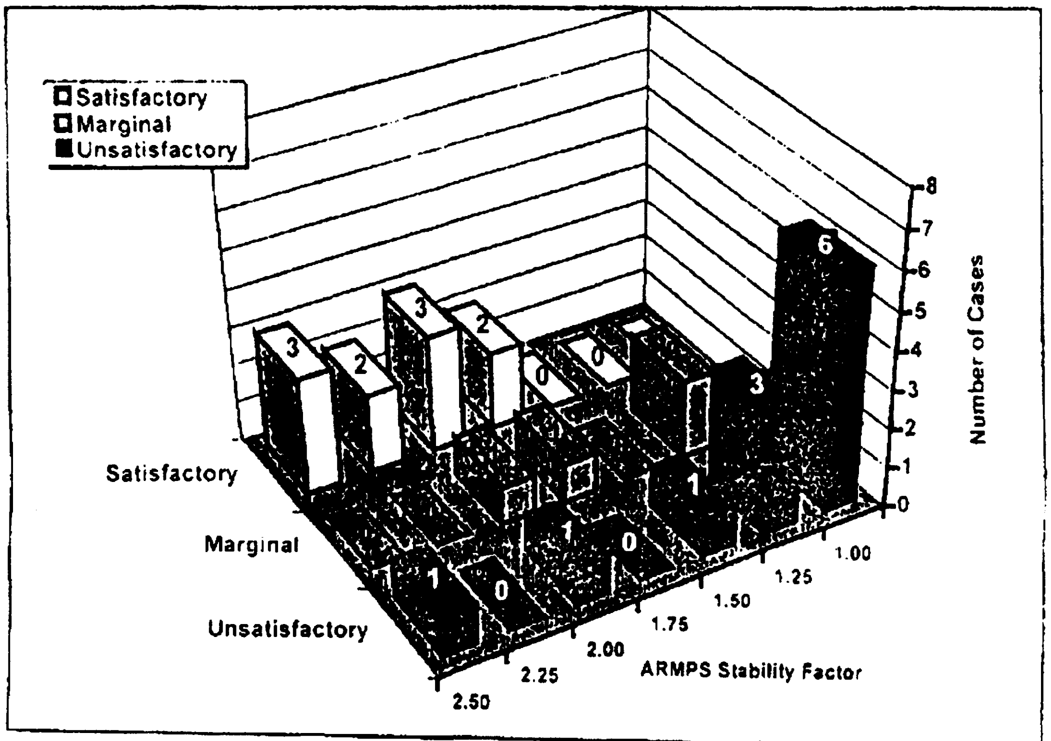


Figure 11.—Sandstone roof case histories from mining complex in southern West Virginia.

GUIDELINES FOR USING ARMPS

ARMPS appears to provide good first approximations of the pillar sizes required to prevent pillar failure during retreat mining. In an operating mine, past experience can be incorporated directly into ARMPS. ARMPS stability factors can be back-calculated for both successful and unsuccessful areas. Once a minimum ARMPS SF has been shown to provide adequate ground conditions, that minimum should be maintained in subsequent areas as changes occur in the depth of cover, coal thickness, or pillar layout. In this manner, ARMPS can be calibrated using site-specific experience.

ARMPS is also well suited for initial feasibility studies where no previous experience is available. Operators may begin with an SF near 1.5, then adjust as they observe pillar performance. ARMPS may also help in optimizing panel designs by identifying pillars that might be needlessly oversized.

ARMPS may be used to analyze a wide variety of mining geometries. For example, most bleeder designs can be analyzed by selecting loading condition 3, then setting the extent of the active gob to zero. The "Help" text included with version 4.0 of the program contains many tips on selecting the proper input parameters when using ARMPS.

In some cases, more detail may be desired than can be provided by ARMPS. Some complex situations, such as multiple-seam interactions, are beyond the capabilities of ARMPS. In these instances, the newly developed LAMODEL [Heasley 1997] may be the appropriate tool to use.

CONCLUSIONS

The ARMPS program has already proven to be a useful aid in planning pillar recovery operations. It is easy to use, and a large number of analyses can be run in a relatively short period. The program is sufficiently flexible to be applicable to a wide variety of mining geometries. If the user desires, it also provides a full range of intermediate calculations in addition to the SF. Many mines throughout the United States and abroad already use ARMPS, and the Mine Safety and Health Administration has also made extensive use of the program.

Current efforts are aimed at improving the interpretation of the ARMPS SF. Although pillar failures seem unlikely when the ARMPS SF is greater than 1.5, there are apparently many cases where SF values as low as 0.75 have been successful. Factors such as roof quality, floor strength, and mining method may determine whether a pillar design succeeds. These factors are now being included in the retreat mining case history data base and will be integrated into future design guidelines.

To obtain a single copy of the ARMPS computer program, version 4.0 for Windows, send three double-sided, high-density diskettes to: Christopher Mark, Ph.D., NIOSH, Pittsburgh Research Center, Cochrans Mills Rd., P.O. Box 18070, Pittsburgh, PA 15236-0070.

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APPENDIX A.—DERIVATION OF THE MARK-BIENIAWSKI PILLAR STRENGTH FORMULA

Early versions of the ARMPS program, following the ALPS program, used the Bieniawski formula to estimate pillar strength [Bieniawski 1992]:

$$S_p = S_1 [0.64 + (0.36\ w/h)], \qquad \text{(A-1)}$$

where S_p = pillar strength,

S_1 = in situ coal strength,

w = pillar width (or least plan dimension),

and h = pillar height.

The Bieniawski formula was originally developed in the 1960's from in situ testing of large-scale coal specimens. The specimen strengths were determined as the ultimate load-bearing capacity divided by the area. Bieniawski recognized that the formula underestimated the strength of rectangular pillars, however, because all of the specimens were square, there was no obvious way of estimating a "pillar length" effect.

It has been recognized that a major disadvantage of empirical formulas, like that of Bieniawski, is that they treat the pillar as a single structural element. In reality, the stress within even a relatively small pillar is highly nonuniform. Tests conducted by Wagner [1974] demonstrated this quite dramatically (figure A-1).

Modern *mechanics-based* approaches to coal pillars begin with stress distribution. Perhaps the best known is the approach proposed by Wilson [1973, 1983]. Wilson derived an expression for the vertical stress gradient within the yield zone, which he then integrated over the area of the pillar (figure A-2) to determine the ultimate pillar resistance (R). The "pillar strength" is simply the ultimate pillar resistance divided by the pillar area. Numerical models also provide stress distribution profiles, although not normally in the form of an equation. Mechanics-based approaches can be used to evaluate any pillar shape, because the stresses within the pillar are determined by laws that are independent of overall pillar geometry.

Although empirical formulas do not explicitly consider the effect of internal pillar mechanics, it is apparent that they imply a nonuniform stress distribution because of the shape effect. Once the implied stress gradient has been derived, the length effect can be readily determined. The derivation has been published previously [Mark et al. 1988; Mark and Iannacchione 1992] and is summarized below.

First, three assumptions are implicit in Wilson's and other analytical formulations:

1. The stress within the yield zone of a given pillar is a continuous function of the distance from the nearest rib.

2. The stress gradient within the yield zone of a given pillar does not change with time or load (i.e., the yielded coal is perfectly plastic).

3. The stress distribution is symmetric with respect to the center of the pillar.

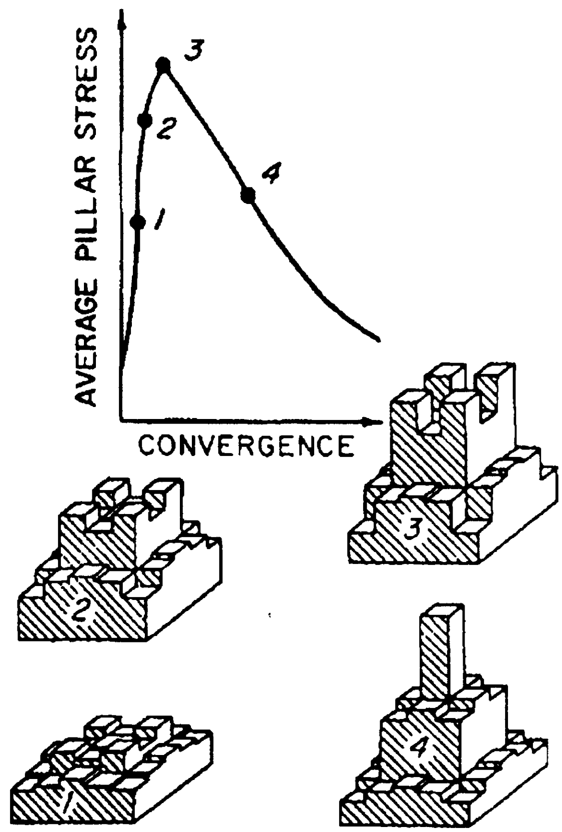

Figure A-1.—Pillar stress profiles measured in small coal pillars (after Wagner [1974]).

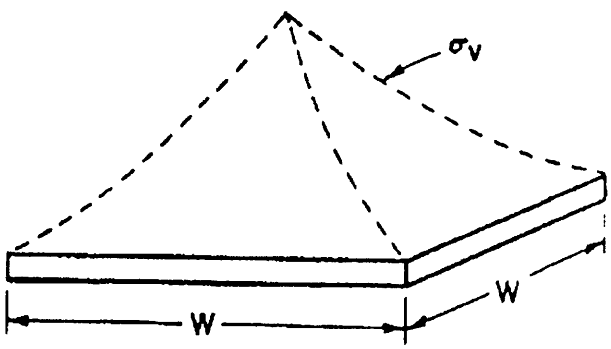

Figure A-2.—Determination of pillar load-bearing capacity as the integral of the pillar stress distribution.

The next step in the derivation is to calculate the ultimate resistance of a square pillar. Using the Bieniawski formula:

$$R = S_1 \left(0.64 + 0.36 \frac{w}{h}\right) w^2. \tag{A-2}$$

Then, the increase in pillar resistance dR due to an increase in cross-sectional area $dA = 2w\ dw$ (figure A-3*A*) may be calculated by taking the derivative of equation A-2 with respect to w:

$$dR = S_1 \left(1.28 + 1.08 \frac{w^2}{h}\right) dw. \tag{A-3}$$

In the next step, the assumption that the vertical pillar stress is a continuous function of the rib distance (x) is applied. It may be seen (figure A-3*B*) that

$$dR = 4 \int_0^{\frac{w}{2}} \sigma_v\ dx\ dw. \tag{A-4}$$

Equating A-3 and A-4 and simplifying, we have

$$S_1 = \left(0.32\ w + 0.27 \frac{w^2}{h}\right) = \int_0^{\frac{w}{2}} \sigma_v\ dx. \tag{A-5}$$

The function that satisfies equation A-5 is

$$\sigma_v = S_1 \left(0.64 + 2.16 \frac{x}{h}\right). \tag{A-6}$$

Equation A-6 is the stress gradient in the yield zone predicted by the Bieniawski formula. Stress gradients have also been derived for several other common empirical pillar strength formulas [Mark and Iannacchione 1992].

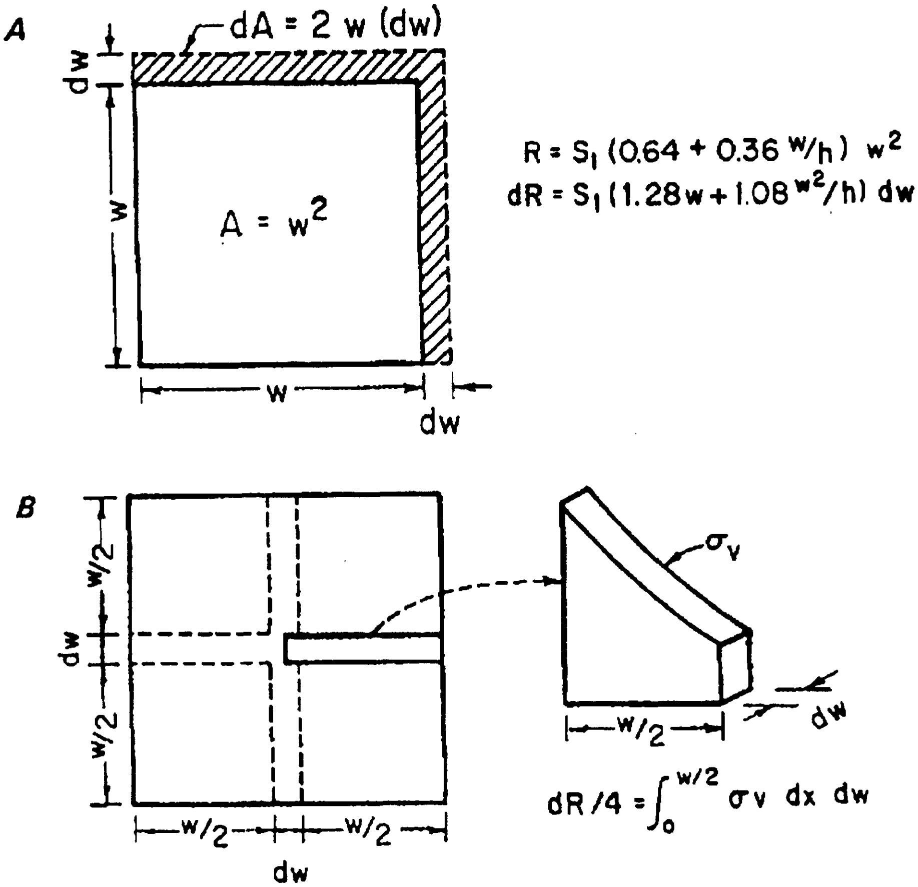

Figure A-3.—Determination of pillar stress gradients from a pillar strength formula. *A*, calculation of dR directly from the formula; *B*, calculation of dR in terms of the vertical stress gradient.

To determine the load-bearing capacity of any pillar shape, it is now only necessary to integrate equation A-6 over the load-bearing area of the pillar. For example, the load-bearing capacity of an extremely long strip pillar (R_s) is

$$R_s = 2L\int_0^{\frac{w}{2}} S_1 \left(0.64 + 2.16\frac{x}{h}\right) dx. \tag{A-7}$$

Solving:
$$R_s = (Lw)\, S_1 \left(0.64 + 0.54\,\frac{w}{h}\right). \tag{A-8}$$

Dividing by the pillar area (Lw) yields the strength of a strip pillar (S_s):

$$S_s = S_1 \left(0.64 + 0.54\,\frac{w}{h}\right). \tag{A-9}$$

Equation A-9 implies that a strip pillar's strength can approach 150% that of a square pillar, but that the strength difference is reduced as the w/h ratio is reduced.

The ultimate load carried by a rectangular pillar is equivalent to the load carried by a square pillar of width w plus a section of a strip pillar of length (L − w), as shown in figure A-4. Combining equations A-6 and A-9, the ultimate load carried by a rectangular pillar (R_r) is

$$R_r = S_1 \left\{\left[w^2\left(0.64 + 0.36\,\frac{w}{h}\right)\right] + \left[(w(L-w))\left(0.64 + 0.54\,\frac{w}{h}\right)\right]\right\}. \tag{A-10}$$

Simplifying:

$$R_r = S_1 \left[0.64\, wL + 0.54\left(w^2\,\frac{L}{h}\right) - 0.18\left(\frac{w^3}{H}\right)\right]. \tag{A-11}$$

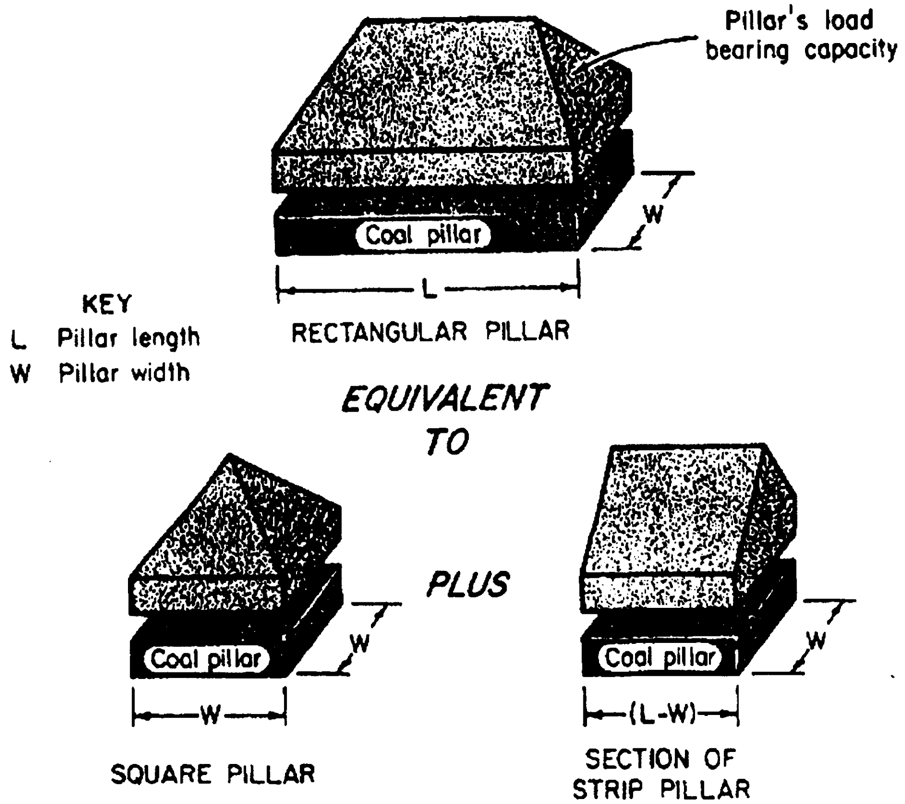

Figure A-4.—Pillar stress distributions for square, strip, and rectangular pillars.

Dividing by the load-bearing area (wL), the Mark-Bieniawski formula is obtained

$$S_p = S_1 \left[0.64 + 0.54\left(\frac{w}{H}\right) - 0.18\ \frac{w^2}{Lh}\right]. \qquad \text{(A-12)}$$

Equation A-12 indicates that the increase in strength in a rectangular pillar depends on both (w/h) and (w/L). Table A-1 compares the pillar strengths determined by the Mark-Bieniawski formula with those obtained from the Bieniawski formula.

Table A-1.—Pillar strength from the Mark-Bieniawski formula, assuming the strength of a square pillar (original Bieniawski formula) as unity

Pillar L/w	Pillar w/h				
	1	2	4	10	20
1.5	1.06	1.09	1.12	1.14	1.16
2.0	1.09	1.13	1.18	1.21	1.23
4.0	1.14	1.23	1.32	1.41	1.45
10.0	1.16	1.25	1.34	1.42	1.46

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APPENDIX B.—ARMPS CASE HISTORY DATA BASE

Table B-1.—Unsatisfactory pillar retreat case histories

State and coal seam	ARMPS SF	Seam thickness, m (ft)	Depth, m (ft)	Loading condition
Alabama:				
Blue Creek	1.54	1.8 (6.0)	350 (1,150)	2
Blue Creek	0.99	1.8 (6.0)	350 (1,150)	3
Colorado:				
Cameo	0.74	2.1 (7.0)	90 (300)	1
D	1.20	2.7 (9.0)	260 (850)	2
D	0.99	2.7 (9.0)	305 (1,000)	3
Kentucky:				
Harlan	1.16	3.7 (12)	285 (940)	1
Harlan	0.96	2.1 (7.0)	305 (1,000)	1
Harlan	0.86	3.7 (12)	260 (850)	2
Harlan	1.12	3.7 (12)	325 (1,070)	1
Hazard No. 4	0.44	3.0 (10)	305 (1,000)	4
Hazard No. 4	0.56	1.3 (4.2)	245 (800)	3
Hazard No. 4	0.50	1.5 (5.0)	215 (700)	3
Lower Elkhorn (No. 2 Gas)	1.03	4.0 (13.0)	245 (800)	1
Lower Elkhorn (No. 2 Gas)	1.02	4.0 (13.0)	185 (600)	3
Ohio:				
Lower Freeport	1.20	1.5 (5.0)	215 (700)	1
Mahoning	0.66	1.0 (3.3)	75 (250)	1
Mahoning	0.95	1.0 (3.3)	75 (250)	1
Pennsylvania:				
Lower Kittanning	1.41	2.0 (6.5)	115 (380)	2
Lower Kittanning	1.55	2.0 (6.5)	120 (400)	2
Lower Kittanning	1.29	2.1 (7.0)	75 (250)	1
Pittsburgh	0.97	2.1 (7.0)	275 (900)	3
Pittsburgh	1.17	2.3 (7.5)	150 (500)	3
Pittsburgh	1.29	2.2 (7.2)	245 (810)	4
Pittsburgh	1.15	2.2 (7.2)	245 (810)	4
Sewickley	1.82	1.6 (5.25)	185 (600)	3
Tennessee:				
Beach Grove	1.26	0.8 (2.5)	315 (1,025)	1
Beach Grove	0.88	0.8 (2.5)	305 (1,000)	3
Utah:				
Blind Canton	0.84	2.5 (8.3)	365 (1,200)	3
Gilson	0.76	2.7 (9.0)	365 (1,200)	3
Gilson	0.43	2.7 (9.0)	515 (1,690)	3
Lower O'Connor	0.95	5.3 (17.5)	170 (550)	1
Virginia:				
Blair	1.37	1.2 (3.8)	185 (600)	3
Glamorgan	1.06	1.8 (6.0)	215 (700)	3
Jawbone	1.53	1.3 (4.2)	215 (700)	3
Jawbone	1.47	1.4 (4.6)	150 (500)	3
Pocahontas No. 3	0.61	1.7 (5.5)	520 (1,700)	1
Pocahontas No. 3	1.35	1.5 (5.0)	150 (500)	3
Pocahontas No. 4	1.03	2.4 (8.0)	90 (300)	1
West Virginia:				
Beckley	0.72	1.8 (6.0)	350 (1,150)	4
Coalburg	0.75	2.4 (8.0)	90 (300)	1
Coalburg	0.59	2.7 (9.0)	120 (400)	NAp
Coalburg	0.98	2.7 (9.0)	120 (400)	NAp
Coalburg	1.10	2.7 (9.0)	120 (400)	NAp
Coalburg	1.35	2.7 (9.0)	120 (400)	NAp

See explanatory notes at end of table.

Table B-1.—Unsatisfactory pillar retreat case histories—Continued

State and coal seam	ARMPS SF	Seam thickness, m (ft)	Depth, m (ft)	Loading condition
West Virginia —Continued				
Dorothy	1.36	3.7 (12.0)	95 (315)	3
Dorothy	1.37	3.7 (12.0)	95 (315)	2
Dorothy (Winifrede)	1.15	3.4 (11.0)	70 (225)	1
Dorothy (Winifrede)	1.45	3.4 (11.0)	70 (225)	4
Dorothy (Winifrede)	1.39	3.7 (12.0)	95 (315)	2
Dorothy (Winifrede)	1.02	3.0 (10.0)	55 (175)	1
Dorothy (Winifrede)	1.15	3.0 (10.0)	100 (325)	2
No 2 Gas	0.95	1.4 (4.5)	245 (800)	4
Stockton	0.84	3.0 (10.0)	70 (225)	2
Stockton	0.96	3.0 (10.0)	75 (240)	1
Stockton	0.82	3.0 (10.0)	75 (245)	1
Stockton	1.47	3.0 (10.0)	85 (280)	1
Stockton	1.19	3.0 (10.0)	85 (280)	2
([1])	0.72	1.5 (5.0)	120 (400)	1
([1])	0.82	1.4 (4.5)	115 (375)	1

NAp Not applicable.

[1]Not provided by original reference.

Table B-2.—Satisfactory pillar retreat case histories

State and coal seam	ARMPS SF	Seam thickness, m (ft)	Depth, m (ft)	Loading condition
Alabama:				
Blue Creek	1.96	1.8 (6.0)	350 (1,150)	2
Colorado:				
Cameo	1.86	2.1 (7.0)	120 (400)	3
Cameo	1.14	2.1 (7.0)	215 (700)	2
Cameo	0.93	2.1 (7.0)	245 (800)	3
D	1.23	2.7 (9.0)	260 (850)	2
D	1.44	2.7 (9.0)	215 (700)	2
Illinois:				
Herrin No. 6	1.14	2.4 (8.0)	215 (700)	3
Kentucky:				
Harlan	1.94	2.0 (6.5)	90 (300)	3
Hazard No. 4	1.36	1.3 (4.4)	130 (420)	3
Kellioka	1.41	1.5 (5.0)	260 (860)	2
Kellioka	1.18	1.5 (5.0)	205 (675)	3
Kellioka	0.45	1.5 (5.0)	440 (1,450)	3
Kellioka	1.61	1.5 (5.0)	185 (600)	3
Lower Elkhorn (No. 6 Gas)	1.64	4.0 (13.0)	120 (400)	3
Pond Creek	1.20	1.7 (5.5)	215 (700)	2
Pond Creek	1.70	1.7 (5.5)	135 (450)	3
Pond Creek	2.0	1.7 (5.5)	120 (400)	2
Pond Creek	1.98	1.7 (5.5)	135 (450)	3
Pond Creek	1.69	1.7 (5.5)	135 (450)	2
Ohio:				
Lower Freeport	1.80	1.5 (5.0)	170 (550)	1
Lower Freeport	1.70	1.5 (5.0)	170 (550)	1
Mahoning	2.50	1.0 (3.3)	75 (250)	1
Pennsylvania:				
Lower Freeport	2.06	1.8 (6.0)	120 (400)	3
Lower Kittanning	1.65	2.0 (6.5)	115 (380)	3
Lower Kittanning	1.78	2.0 (6.5)	115 (380)	3
Lower Kittanning	1.79	2.0 (6.5)	120 (400)	3
Lower Kittanning	1.85	2.0 (6.5)	120 (400)	2
Lower Kittanning	2.14	1.5 (5.0)	170 (550)	3
Pittsburgh	1.89	2.1 (7.0)	150 (500)	3
Pittsburgh	2.78	2.2 (7.2)	260 (855)	2
Sewickley	1.70	1.6 (5.25)	185 (600)	3
Sewickley	2.32	1.6 (5.25)	185 (600)	2
Upper Freeport	1.88	1.3 (4.2)	65 (210)	1
Tennessee:				
Beach Grove	0.98	0.8 (2.5)	315 (1,025)	2
Utah:				
Gilson	0.50	2.7 (9.0)	610 (2,000)	2
Virginia:				
Blair	1.65	1.2 (3.8)	185 (600)	3
Glamorgan	2.31	1.8 (6.0)	120 (400)	3
Jawbone	2.86	1.3 (4.2)	135 (450)	2
Jawbone	2.15	1.3 (4.2)	150 (500)	3
Jawbone	1.97	1.4 (4.6)	120 (400)	3
Mossy-Haggy	2.05	0.9 (3.0)	150 (500)	3
Pocahontas No. 3	0.92	1.7 (5.5)	520 (1,700)	2
Pocahontas No. 3	1.21	1.7 (5.5)	520 (1,700)	3
Pocahontas No. 3	1.89	1.5 (5.0)	150 (500)	2
Pocahontas No. 4	0.91	1.8 (6.0)	365 (1,200)	3
Pocahontas No. 4	2.77	0.9 (3.0)	90 (300)	2
Pocahontas No. 4	0.76	2.0 (6.5)	440 (1,450)	3
Red Ash	2.44	0.9 (3.0)	150 (500)	2
Red Ash	2.44	0.9 (3.0)	215 (700)	3
Tiller	2.22	1.2 (4.0)	150 (500)	3

See explanatory notes at end of table.

Table B-2.—Satisfactory pillar retreat case histories—Continued

State and coal seam	ARMPS SF	Seam thickness, m (ft)	Depth, m (ft)	Loading condition
West Virginia:				
Beckley	0.90	1.8 (6.0)	350 (1,150)	4
Beckley	1.17	2.7 (9.0)	260 (850)	4
Coalburg	1.14	2.7 (9.0)	120 (400)	NAp
Coalburg	1.30	2.7 (9.0)	120 (400)	NAp
Coalburg	1.41	2.7 (9.0)	120 (400)	NAp
Coalburg	1.50	2.7 (9.0)	120 (400)	NAp
Coalburg	1.59	2.7 (9.0)	120 (400)	NAp
Coalburg	1.76	2.7 (9.0)	120 (400)	NAp
Coalburg	1.91	2.7 (9.0)	120 (400)	NAp
Coalburg	2.17	2.7 (9.0)	120 (400)	NAp
Coalburg	2.37	2.7 (9.0)	120 (400)	NAp
Coalburg	2.41	2.7 (9.0)	120 (400)	NAp
Dorothy (Winifrede)	2.10	3.4 (11.0)	70 (225)	2
Dorothy (Winifrede)	1.32	3.0 (10.0)	85 (285)	2
Dorothy (Winifrede)	1.49	3.0 (10.0)	100 (325)	2
Dorothy (Winifrede)	1.72	3.0 (10.0)	70 (225)	2
Fire Creek	1.24	1.4 (4.5)	260 (850)	2
Lower Winifrede	1.73	2.0 (6.5)	185 (600)	2
Peerless	1.56	1.4 (4.75)	215 (700)	2
Sewell	2.55	1.2 (4.0)	105 (350)	2
Stockton	1.56	3.0 (10.0)	65 (220)	2
Stockton	1.99	3.0 (10.0)	75 (245)	2

NAp Not applicable.

MODELING OF CASTLEGATE SANDSTONE ESCARPMENT STABILITY

Prepared for

C.W. MINING COMPANY

Huntington, Utah

July 2001

By

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1. INTRODUCTION

This report with attachments was prepared in response to a request by C.W. Mining Co. that Maleki Technologies, Inc. (MTI), complete an analysis of the stability of the Castlegate Sandstone escarpment for two-seam mining in the Wild Horse Ridge reserves. The specific objectives and the scope of work were outlined by C.W. Mining and consisted of : (1) evaluate surface subsidence and the stability of the Castlegate Sandstone escarpments and (2) analyze the distances that unstable material may travel after mining.

C.W. Mining Co has used the room-and-pillar technique to extract three-seam reserves in coal fields on the Wasatch Plateau near Huntington, Utah (figure 1). These seams are located toward the base of the Blackhawk Formation and consist of the Tank, Blind Canyon, and Hiawatha. Within the present two-seam mining area (to the west of Bear Canyon), the Tank Seam and the Blind Canyon Seam are locally mined in descending and ascending order. Mining has taken place in conjunction with geotechnical monitoring in two panels in each seam (Maleki and others 1999, 2000). Because ground conditions between the present mining areas and the Wild Horse Ridge reserve are similar, existing data were used to estimate potential recovery and associated ground movement in the Wild Horse Ridge reserve. In addition, longwall mining experience and geotechnical data regarding mining in the vicinity of escarpments near mines operated by Energy West are available (Jones 1994; Maleki and others 2000). Attachment A contains technical papers dealing with geotechnical and escarpment assessments in the C.W. Mining and neighboring mines.

During the last decade, Energy West Mining Co., government agencies, academia, and MTI have directed a significant amount of research to developing predictive tools for assessing the stability of the Castlegate Sandstone escarpment. These studies were initiated to satisfy needs for escarpment stability and resource recovery. Escarpment instability is associated with the release of rocks from the escarpment face onto slopes below the escarpment. The government agencies responsible for the administration and protection of nonmineral resources in the area (U.S. Forest Service; Fish and Wildlife Service; Utah Division of Wildlife Resources; Utah Division of Oil, Gas, and Mining) are interested in the impact of escarpment failure on animal and plant life and public safety, The Bureau of Land Management (BLM), which administers and leases mineral resources, is responsible for ensuring that mining companies maximize resource recovery. Thus, the optimum goal has become one of meeting the requirement of resource recovery while minimizing mining impacts on the surrounding environment.

To achieve this goal, interested parties have recently focused on development of a predictive statistical technique so that the economic benefits of mining can be analyzed against the probability of escarpment failure and its influence on the surrounding environment. In cooperation with Energy West engineers, MTI has been developing this statistical model to assess the impacts of longwall mining on Corncob Wash and Newberry Canyon while characterizing escarpment conditions along Rilda Canyon reserves in detail. The model will be fine tuned after mining in Rilda Canyon is complete in the near future (appendix A). C.W. Mining's experience with room-and-pillar techniques will complement the existing escarpment stability database by allowing an evaluation of both longwall and room-and-pillar mining methods.

To evaluate escarpment stability in C.W. Mining operations, we have expanded the existing models for Energy West mines to typical sections in the Wild Horse Ridge while accounting for differences in geologic and mining conditions. Two known factors that are different in the Wild Horse Ridge reserve area are (1) use of room-and-pillar mining to the south near most of the escarpments instead of longwall mining and (2) the shorter distance between the sandstone escarpments and the Tank Seam. Room-and-pillar mining is associated with 50% less subsidence and thus is beneficial to escarpment stability. The shorter distance between the Tank Seam and the Castlegate Sandstone is detrimental. Overall, these two factors cancel out and the existing models and data from neighboring mines are judged to be applicable for this preliminary evaluation until more detailed monitoring data become available.

To the south where escarpments of interest to this study are exposed (figure 2), C.W. Mining is planning to extract the Tank and Blind Canyon seams using the room-and-pillar mining technique in a descending order. Two-seam longwall mining is planned to the north for the extraction of the Tank and Hiawatha seams. This longwall area is not overlain by Castlegate escarpments in most areas, excepting reserves within the initial four longwall panels. The escarpment study area consists of the eastern side of Bear Canyon and the areas located near the Left Fork of Fish Creek.

In response to a request by C.W. Mining, MTI has used its models and comparative analysis techniques to address escarpment stability along the Wild Horse Ridge study area for permitting purposes. C.W. Mining has the ultimate responsibility for analyzing and mitigating any safety issues related to escarpment instability. MTI does not offer any expressed or implied warranty with regard to public safety for any option selected and implemented by C.W. Mining for controlling ground movements.

The scope of this work consists of (1) a review of the geotechnical framework for the Wild Horse Ridge reserve, (2) a review of subsidence characteristics for two-seam reserves in western U.S. mines, (3) an evaluation of escarpment stability, and (4) estimates of the distances rocks travel.

Following the introduction, conclusions and recommendations are given in section 2, and the geotechnical framework for the C.W. Mining reserve is analyzed in section 3. Section 4 presents a description of subsidence characteristics in western U.S. mines and identifies typical subsidence factors and the angle of draw for C.W. Mining's geologic conditions. The stability of the Castlegate Sandstone escarpment is evaluated in section 5, and possible rock travel distances are discussed in section 6. Travel distances are estimated along a cross section of interest using the Colorado Rock Fall Simulation program and site-specific data from failures at Energy West and C.W. Mining operations.

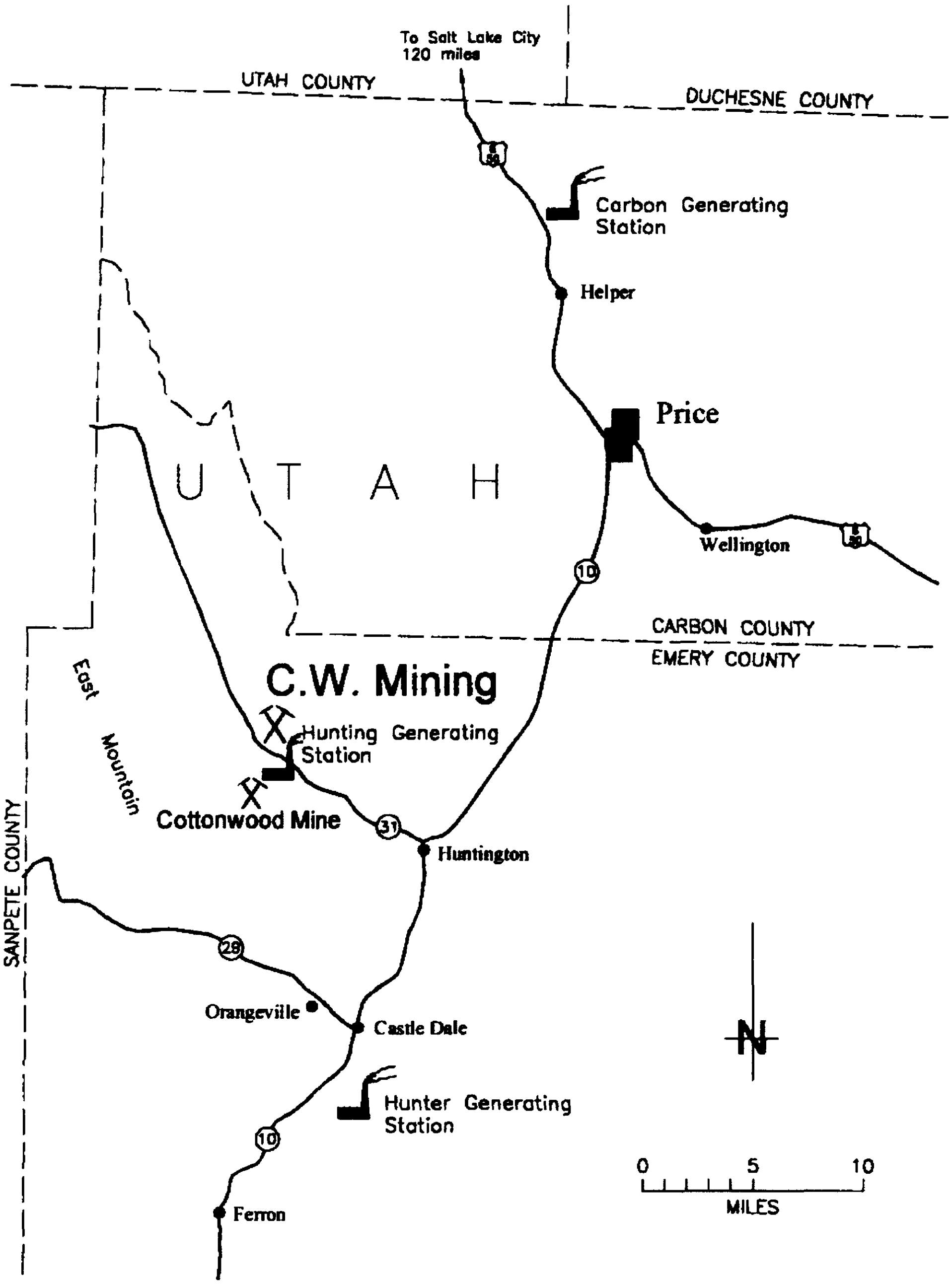

Figure 1. Project location map.

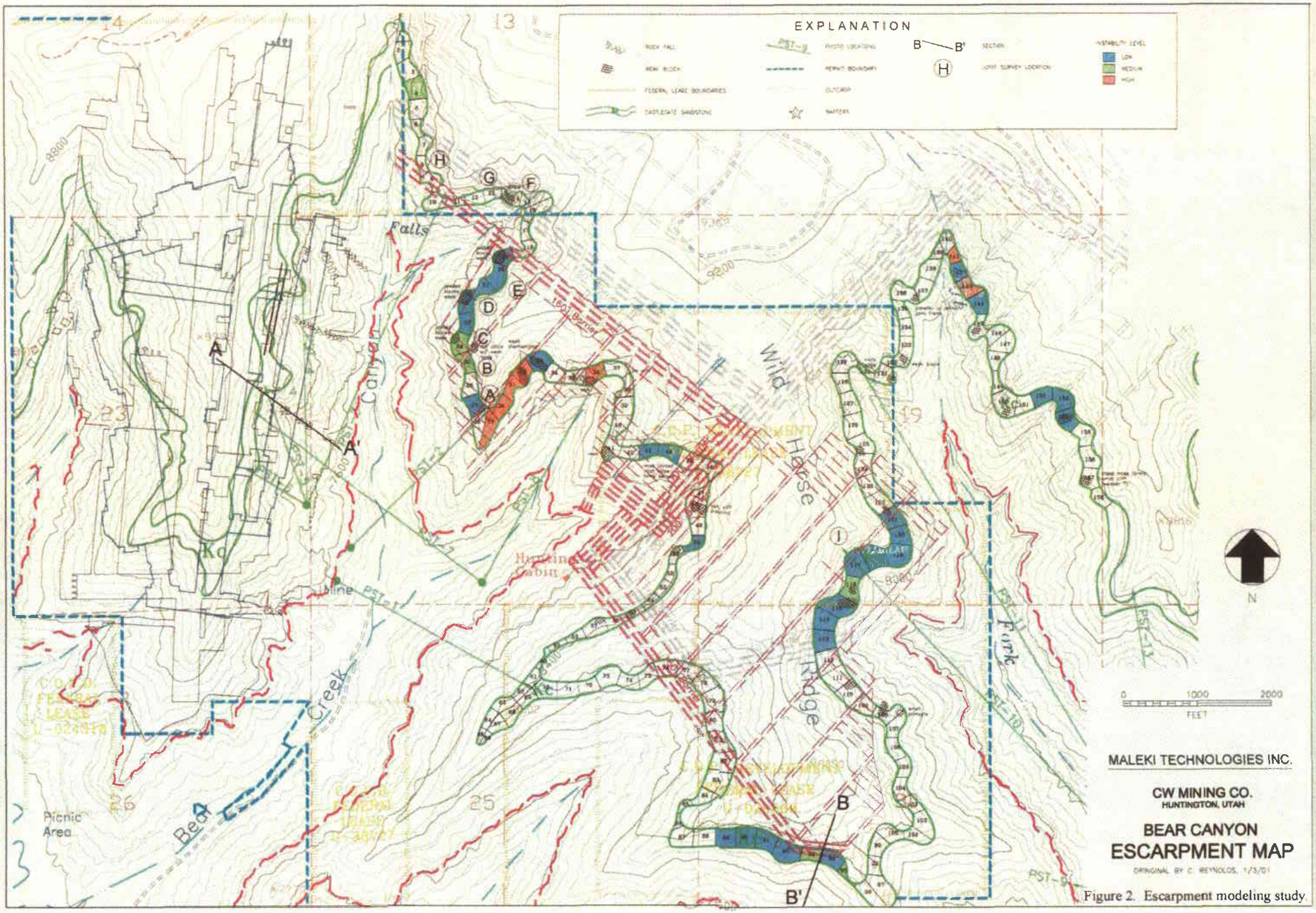

Figure 2. Escarpment modeling study

2.0 SUMMARY AND CONCLUSIONS

In response to a request by C.W. Mining, MTI evaluated escarpment stability along the Wild Horse Ridge study area using the Rockrisk computer program. Travel distances along a cross section of interest were estimated using rock fall simulation programs and experience in the area. To assess escarpment stability, the area of interest was divided into 158 study cells. For each cell, escarpment and canyon geometries were characterized using base maps provided by C.W. Mining and field observations. An instability index was calculated using the Rockrisk computer program and decades of experience in the mine area.

Three instability levels indicating overall low to medium volumes of spalling are identified in figure 2. These instability levels are—

- Low (not likely to have significant spalling)
- Medium (likely to have spalling)
- High (very likely to have significant spalling).

Rock fall simulations along cross sections of interest are in general agreement with observations of mining-induced rock travel distances of 800 ft in the neighboring canyons. Maximum rock travel distances, however, vary according to canyon slope and frictional and rebound properties of rock blocks.

It is recommended that escarpment stability be monitored selectively, depending on safety and environmental needs. The data from such monitoring will also be important for verifying the assumptions of this study and for enhancing model capabilities when predicting the stability of the Castlegate Sandstone escarpment during multiple-seam room-and-pillar mining.

3.0 GEOTECHNICAL FRAMEWORK

3.1 Stratigraphy

The three coal seams of economic interest belong to the Blackhawk Formation, which is overlain by the Castlegate Sandstone and underlain by the Star Point Sandstone and the Mancos Shale (figure 3). Minable seams are the Tank and Blind Canyon to the south and the Tank and Hiawatha to the north of the Wild Horse Ridge reserves. These seams average 8, 11, and 6 to 8 ft in the study area .

The overlying, cliff-forming Castlegate Sandstone is a massive cross-bedded unit. It contains intermittent thin interbeds of shale, pebble conglomerate, and mudstone. Based on corehole data, this unit is 170 to 250 ft thick in the area; however, the actual exposed thickness is much less locally (as low as 50 ft). The exposed thickness of the Castlegate Sandstone escarpment is directly related to the stability of the escarpment and the volume of displaced rocks (Maleki and others 2000). The Price River Formation consists of numerous beds of cross-bedded sandstones with occasional interbeds of shale, pebble conglomerate, and mudstone.

The Blackhawk Formation is composed of interbedded deltaic mudstone and siltstone and is less resistant to weathering than the neighboring units. It is characterized by alternating slope- and cliff-forming units. This unit is approximately 750 ft thick.

The Star Point Sandstone consists of thick cliff-forming sandstone units separated by shales. It is light colored and is approximately 350 ft thick in the Wild Horse Ridge area. The Mancos Shale is a blue-grey color marine shale approximately 1,000 ft thick and is soft and well weathered.

3.2 Jointing

A 2-day geologic field mapping trip was conducted by MTI during the course of this study. The objective was to (1) characterize joint orientations in the study area and (2) estimate the exposed thickness of the Castlegate Sandstone. The latter was important in improving the historical data collected by Dwelling regarding the Castlegate Sandstone unit shown on the base map provided by C.W. Mining. During field mapping to the east, C.W. engineering staff collected data regarding the location and amount of failed rocks on the slopes to the west of Bear Canyon over the existing mine. Attachment B is a photographic record of the Castlegate Sandstone escarpment, and figure 2 shows the location of joint measurements (A to I).

The joint patterns at the Castlegate Sandstone horizon are similar across the study area. Joint trends are thought to coincide generally with joints found in the overlying Price River and underlying Blackhawk formations and are consistent with measurements collected on the Wasatch Plateau (Maleki 1988; Maleki and others 1999). Joints were typically within a few degrees from vertical. Three general orientations were identified (table 1).

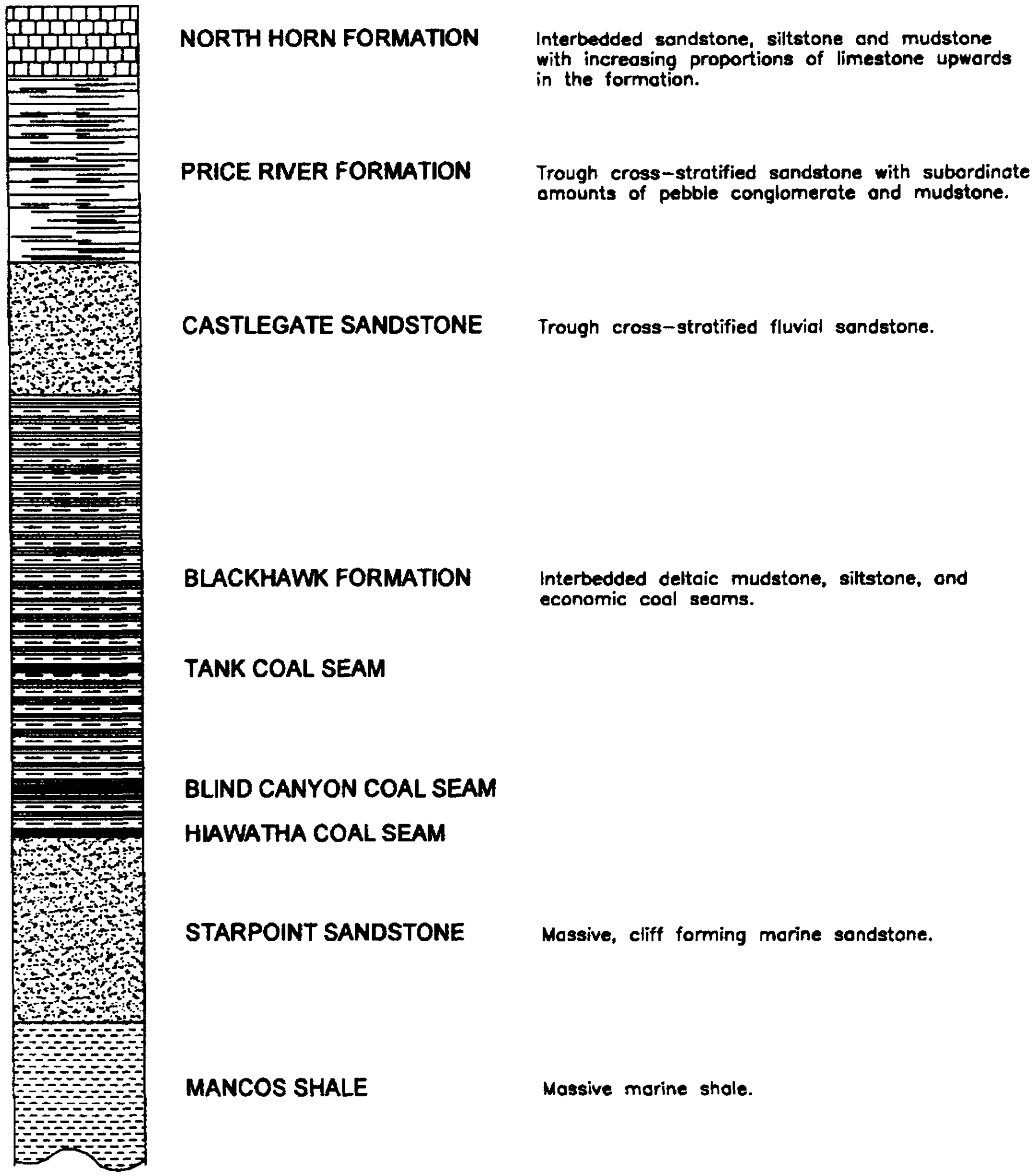

Figure 3. Typical stratigraphic column.

The most pronounced (primary) joint trend typically ranges between N10°E to N20°E (N15°E average). There may be a few degrees of counterclockwise rotation in this joint set as one moves to the east because measurements greater than N20°E were noted only on the west side of Wild Horse Ridge, and measurements less than N10°E were noted only on the east side of the ridge.

A less pronounced, secondary joint system trending S80°E to S90°E was also observed. This trend appeared to be generally consistent across the study area. The only variation in this trend was noted at location 1, but the data are too limited and inclusive.

A third joint set was noted having a N50°E to N55°E trend. This set was only observed in the east near Fish Creek Canyon. Spacing on this set is estimated to be greater than 10 ft due to its lack of occurrence or expression.

Apparent joint spacing appears to be controlled by confining stress. In outcrop, the primary and secondary joints are more apparent and appear more closely spaced at or near the points than in the head of drainages. Rocks in place often exhibit jointing at 10 to 15 ft spacings, but more broken rocks nearly always showed more closely spaced joints.

Table 1. Measured joint orientation and spacing for the Castlegate Sandstone within the Wild Horse Ridge reserve.

Location	Joint set 1	Set 1 spacing, ft	Joint set 2	Set 2 spacing, ft	Joint set 3
A	N16°-25°E	3-6	S75°-85°W	12	
B	N10°-17°E	3-6			
C	N15°-20° E	10-20 subordinated joints at 3			
D	N18°E	5	N85°-90°E	10	
E	N17°-28°E	3-8	S87°E		
F	N18°E	6-8	S82°E	7	
G	N15°E	2-6	S85°E	6-10	
H	N15°-23°E	1-6			
I	N13°-21°E	2-10	S77°-87°E	3-5	N52°E

2.3 Stress Field

Researchers from the former U.S. Bureau of Mines (USBM) and private industry have made a number of stress measurements in mines on the Wasatch Plateau. There are two stress measurements in proximity of C.W. Mining's operations (Maleki and others 1986). These measurements confirm that the far-field stress field is unremarkable. The horizontal stress is moderate and is less than 50% of the vertical stress magnitude. We anticipate a similar stress field at C.W. Mining operations based on observations of the lack of stress-induced stability problems (such as cutters) and an analysis of measurements in the existing reserve (Maleki and others 2000).

3.4 Mechanical Properties

Site-specific geologic and rock mechanics data are limited, although MTI has collected large amounts of information from adjacent properties. The available information on the values for mechanical properties for Blind Canyon coal measure rocks within the existing reserve is summarized in table 2. This data are in general agreement with rock testing results from East Mountain. Jones (1994) reports the results of rock testing and notes uniaxial compressive strengths ranging from 6,000 to 12,000 psi for sandstones and 12,000 to 16,000 psi for shales (mudstones). Similar to the rocks on the C.W. Mining property, the East Mountain units are generally stiff with Young's modulus ranging from 1.6 to 3.4 million psi.

Table 2. Laboratory mechanical properties test results from core testing

Rock type	Uniaxial compressive strength, psi	Young's modulus, million psi	Poisson's ratio
Coal	2,000-3,000	0.4 to 0.45	0.3 to 0.4
Shale	15,000 - 17,000	3 to 4	0.2 to 0.4
Sandstone	7,000-12,000	3 to 4	0.3 to 0.4

4.0 SUBSIDENCE CHARACTERISTICS

4.1 Subsidence Mechanisms

Surface subsidence occurs as a result of downward rock mass movement resulting from closure and collapse of mined-out excavations. Surface subsidence processes result in both vertical and horizontal displacement of rocks. Three major mechanisms of surface subsidence are associated with mining: formation of sinkholes, uniform settling, and formation of troughs. For the C.W. Mining plans using either high-extraction longwall mining or room-and-pillar mining with retreat, we expect trough subsidence.

Trough subsidence is characterized by the formation of a basin. Continuous fracturing from the mine to the surface usually does not occur, and there is much less abrupt surface fracturing than occurs during sinkhole subsidence. Trough subsidence creates an elliptical shape in plan view and extends over large areas, typically from hundreds to thousands of feet in breadth.

There are three subsidence phases associated with trough subsidence. These are shown in figure 4.

- The *subcritical phase* occurs immediately in the beginning when movement is in a small area at the center of the basin.

- The *critical phase* occurs as the basin area expands when the maximum extent of the downward movement is reached at the center. The critical excavation width is generally larger than 1.4 times the overburden thickness and is influenced by position and strength of competent layers within the overburden.

- The *supercritical phase* occurs as the basin develops a flat bottom. In this phase, the basin area continues to increase with the cave area, but subsidence will remain at the maximum extent attained in the critical phase.

Thus, the surface response to longwall mining activity, shown in figure 4, begins with the subcritical phase, then progresses to the critical phase, and finally to the supercritical phase. The subsidence process first shows effects on the surface as the upper strata bend. Tension (expansion) causes near-surface fractures to open and new ones to be created. Figure 4 shows how the middle portion of the excavation expands as subsidence continues, going through a cycle of first tension and then compression, which closes tension cracks. The tighter and more uniform the cave, the better the fracturing and closing process. Final subsidence shows an excavation with the middle portions lower in elevation, but back to a near-original state. Areas on the edge of the excavation basin are subjected to tensile strains.

Various geologic and mining factors affect the extent and magnitude of vertical displacements. The subsidence factor (ratio of total subsidence to mining height) depends on the bulking characteristics of overburden rocks. The angle of draw identifies the limits of vertical displacement beyond the excavation boundaries (figure 4).

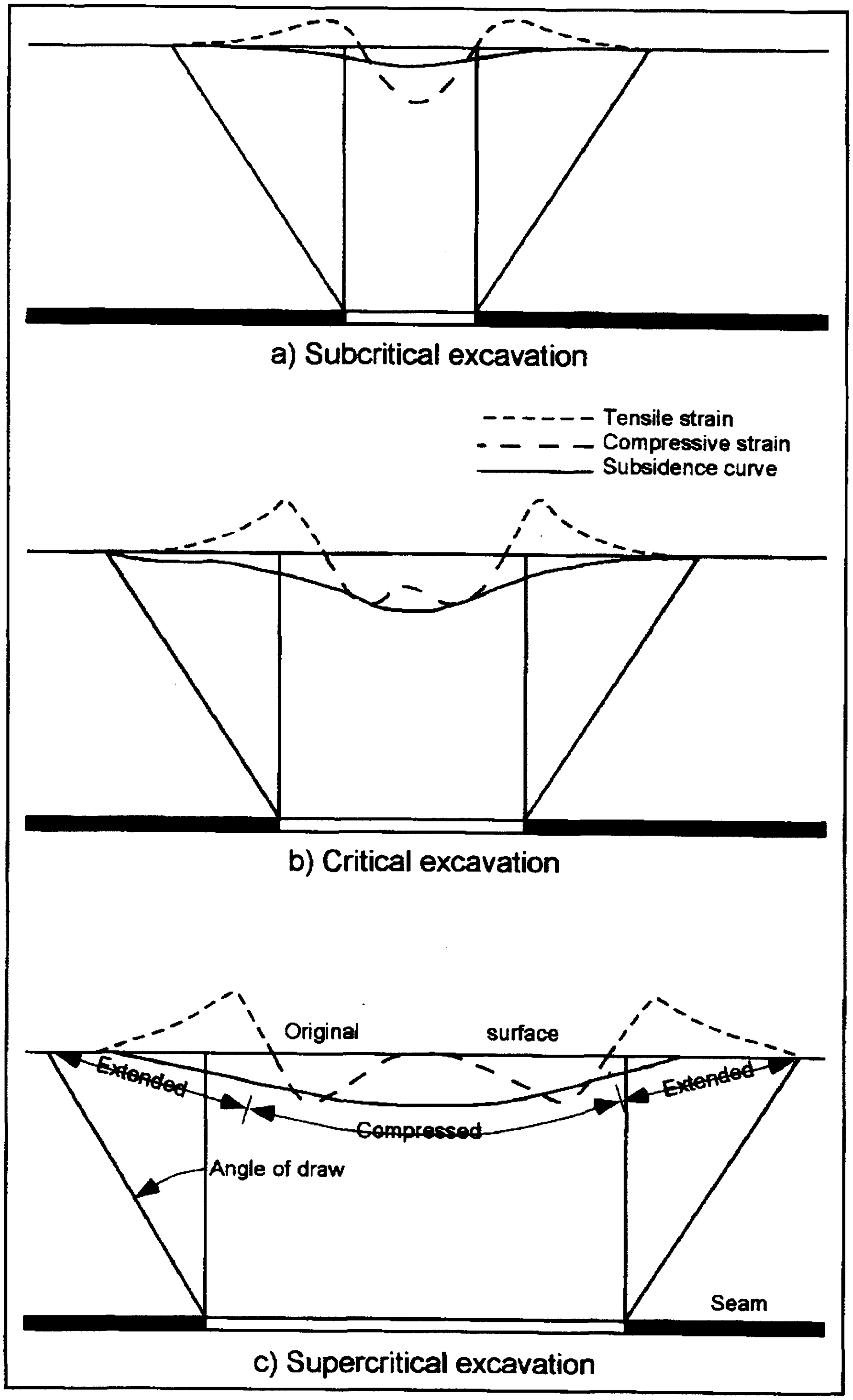

Figure 4. Schematic presentation of subsidence stages as extraction width is increased.

The best estimates for the extent and magnitude of subsidence for the two-seam mining conditions at C.W. Mining can be obtained by reviewing the results of long-term monitoring in Utah. The USBM implemented a comprehensive subsidence study over Energy West's two-seam longwall reserve from 1978 to 1989. The study monitored surface movements over four Blind Canyon and six Hiawatha panels. The study addressed angle of draw, subsidence factors for single and multiple-seam mining, and critical width. Similar to the Wild Horse Canyon reserve, the mining area was bounded by faults. Maximum subsidence was 68% to 72% of the extraction height for single- and two-seam mining conditions, respectively. This is in general agreement with other measurements in Utah that show a subsidence factor of 70%. The angle of draw ranged between 25° to 30° for single- and two-seam mining conditions, respectively. This reported maximum angle of draw is higher than average values for Energy West and other Utah operations (22.5° to 25°, Fejes 1985), but is significantly lower than values reported by the British National Coal Board (NCB) (1975).

Surface subsidence is expected to be limited to an angle of draw of 25° at the Wild Horse Ridge reserve. Maximum subsidence will occur toward the center of the mining area and will range from 5.6 to 13.3 ft for single- and two-seam longwall mining, respectively. For the room-and-pillar mining method, however, maximum subsidence can be much smaller because stumps and pillars are typically left in the gob for ground control purposes. Measurements by researchers from the USBM indicated lower subsidence factors (0.3 to 0.4) (Jones and Kohli 1995; Magers 1993). Applying this subsidence factor to C.W. Mining conditions will result in maximum movement of 2.8 to 6.6 ft under single- and two-seam mining conditions.

Standard subsidence engineering parameters, such as angle of draw and subsidence factor, are useful for estimating expected deformation and for comparing expected ground behavior in different mines under flat topography. An angle of draw of 15° has been used in some situations by the BLM as a simple criterion for balancing the need for resource recovery and protection of sandstone escarpments in Utah mines. Considering the variations in topography over Utah coal mines, the angle of influence is a better indicator of escarpment stability given the geologic and geometric conditions in Utah (Maleki and others 2000) (attachment A). This angle measures the position of the escarpment with respect to full-extraction mining limits (figure 5).

4.2 Rock Movement Associated with Escarpment Instability

Full-extraction mining at the Wild Horse Ridge will reach the supercritical phase with tensile strains forming over the boundaries of mining blocks. For escarpment edges that are near-parallel to panel boundaries and/or joint orientations, instability may result because of the formation of tension cracks at the surface and shear failure at the bottom interface with the Blackhawk Formation. This process contributes to lateral translation of the escarpment, depending on the location of the escarpment with respect to mining limits, escarpment shape (convex or concave), effective height of the escarpment, and the slope of the canyon (Maleki and others 2000). Instability of the escarpment results in release of rocks from the escarpment face onto slopes below the escarpment. The travel distance is influenced by the size of the rocks, surface geometry, and roughness conditions. Travel distances can be calculated using rock fall simulation programs and/or measurements from similar ground.

Two sources provide information on the actual distance traveled by rock on the failure of the Castlegate Sandstone escarpment: (1) extensive measurements taken at Energy West's longwall operations and (2) limited measurements taken near the existing room-and-pillar retreat mining areas to the west of Bear Canyon. Measurements at the Energy West operations indicate that travel distances were approximately 800 ft (Semborski 2000), a distance that is in agreement with estimations at C.W. Mining. In contrast to the Energy West experience, however, the volume of failed rocks is significantly less at the C.W. Mining reserves. This assertion is made based on inspection of escarpment stability along Bear Canyon by C.W. engineering and MTI staff (figure 2) after the completion of mining in the Blind Canyon and Tank seams. Similar to the Energy West experience, instability is confined mostly to concave areas, but the volume of failed rocks is lower.

An estimate of the volume of failed rocks was obtained by C.W. Mining staff using field observations over the existing mine. Estimated volumes for 100-ft-long study cells are between 1 to 10,000 ft^3. This volume reflects changes resulting from mining the Tank Seam. Many areas have been fully undermined in two seams without significant impact on escarpment stability. The lower bench of the Castlegate Sandstone (20 ft high) experienced skin failure during extraction of the Tank Seam. The diameter of the failed rock averaged 2 ft and ranged from 6 in to 6 ft in diameter.

Following are factors that favor the stability of the Castlegate escarpment at the Wild Horse Ridge reserves.

- The mine layout is oriented 45° from the direction of major joints, a factor that is favorable to escarpment stability. This is because mining can activate movements along preexisting joints if they are parallel to panel orientations, particularly at shallow depths.

- The Blind Canyon Seam is significantly thinner than in the existing mine (11 ft on average in comparison to 16 ft). The lower total extraction results in lower subsidence, lower tensile strains, and less potential for escarpment instabilities.

- The planned use of the room-and-pillar mining method near most of the escarpments is beneficial because this method is associated with approximately 50% lower subsidence than full extraction longwall mining (subsidence factor of 0.3-0.4 in comparison to 0.7). The lower subsidence translates into lower surface strain and thus less potential for escapement instability. However since the Tank Seam is locatėd closer to the Castlegate Sandstone, overall, we expect slightly less deformation than calculated for similar conditions in the Energy West longwall operations mining the Blind Canyon Seam.

- The exposed thickness of the Castlegate Sandstone is lower (50 to 200 ft), particularly near Fish Creek Canyon, in comparison to full exposure at neighboring mines.

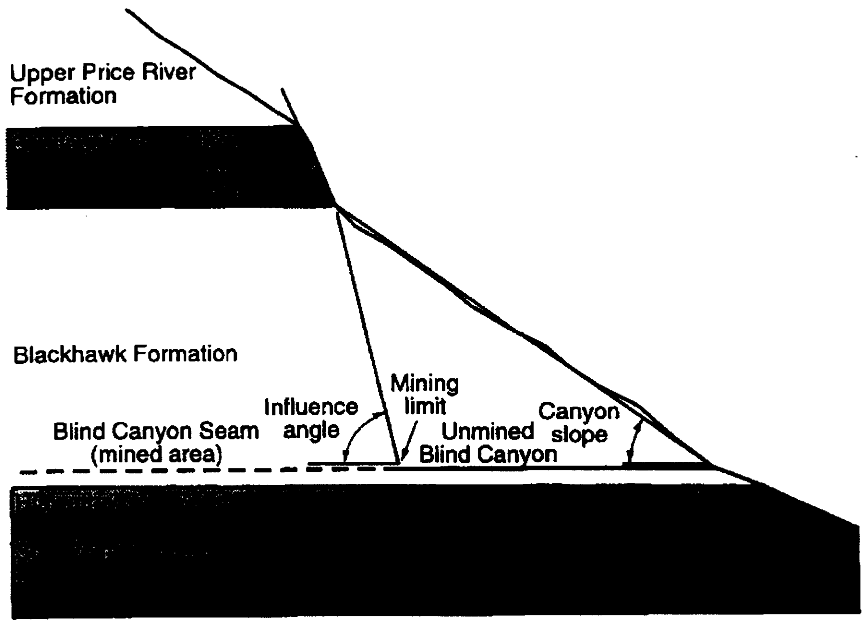

Figure 5. Schematic presentation of pertinent geologic and geometric factors.

5.0 CASTLEGATE SANDSTONE STABILITY

Based on principles of subsidence engineering, geologic investigations, and data available from escapement monitoring on the Wasatch Plateau, we expect ground reactions to two-seam room-and-pillar mining in the C.W. Mining reserves to be similar to the mechanisms studied in the neighboring Energy West mines. Thus, to evaluate Castlegate escarpment stability, we use the model developed for the Energy West operations while using site-specific geologic and geometric conditions obtained for the Wild Horse Ridge reserve.

In this section, we will briefly review the development of this statistical model, identify important variables that influence escarpment stability, and finally, use the model to estimate the potential for escarpment instability in the Wild Horse Reserve study area.

5.1 The Escarpment Stability Model

MTI has used multiple-regression analysis techniques and a wealth of data collected over many years on geology, mining, and escarpment stability to develop this model. An index related to the volume of failed rocks (called the failure index) is used as the response variable after several other factors were considered, including measurements of surface deformation and frequency of mining-induced surface fractures. Geologic and geometric variables were obtained along miles of escarpment exposure at 70 study locations. Regression analysis of data for the first 29 study locations, which had been fully undermined, showed that surface topography (or the shape of the escarpment) played a critical role in influencing escarpment stability.

More recent regression analysis of data from 70 study locations identified several other important geologic and mining factors that influence the stability of the sandstone escarpment. These are—

- Thickness of the Castlegate sandstone
- Angle of mining influence (measuring the position of the escarpment ledge in comparison to full-extraction mining limits), and
- Canyon slope.

The failure index is higher at locations where the canyon slope is steeper (figure 5), the Castlegate exposed thickness is higher, and full extraction is completed beneath and beyond the escarpments. It is interesting to note that joint characteristics are not an influencing factor where an area is fully subsided.

5.2 Escarpment Instability Level

To characterize geologic and mining factors along the Wild Horse Reserve study area, we used field estimates of exposed escarpment thickness, inspected available borehole logs, and used multilayer maps prepared by the C.W. Mining staff (figure 2). The Castlegate escarpment was divided into 158 study cells, each approximately 200 ft wide.

To evaluate escarpment stability, we calculated the failure index for each study cell, given canyon slope, thickness of exposed escarpment, and influence of mining angles. Attachment C presents a histogram frequency diagram for these parameters. Results were summarized in figure 2 in terms of three instability levels:

- Low (not likely to have significant spalling),
- Medium (likely to have spalling), and
- High (very likely to have significant spalling).

Considering the accuracy that can be achieved for calculating geologic and geometric factors using existing overlays and field observations, the proposed instability levels provide a convenient means of assessing the potential and severity of escarpment instability. The areas identified as highly unstable will benefit from monitoring. Monitoring should also be considered at other areas, depending on functional requirements for the canyon and safety and environmental concerns. For instance, we recommend limited monitoring near study cells 78-79 because of a lack of in-depth geotechnical data, among other factors.

6.0 SIMULATED TRAVEL DISTANCES

In section 3.2, we provided a discussion of travel distances for failed rocks using experience from both existing (west Bear Canyon) and neighboring mines. In this section, we will provide estimates of travel distances by using the Colorado Rock Fall Simulation program. For this, we will use the program (1) to simulate rock movements along an existing failure path (section A-A', west Bear Canyon) for calibration of the model and (2) to estimate travel distances along a selected location in the southern Wild Horse Ridge reserve area (cross section B-B', figure 2).

6.1 Background, Colorado Rock Fall Simulation Program

The Colorado Department of Transportation in collaboration with the Colorado School of Mines developed a computer program (CRSP) for simulating rock movement along slopes and canyons. The program is based on detailed observations of rock falls, as well as considerations of the kinematics of a rolling block. To estimate a statistical distribution of travel distances, the program uses canyon profile, rebound, and friction characteristics of the canyon floor and the kinematic energy of a selected number of rocks rolling down the canyon.

The program has been extensively used by the Colorado Department of Transportation, as well as by Energy West, with good success. Based on extensive simulations and back-analyses of travel distances at the Energy West operations, it is possible to estimate travel distances reliably for similar conditions at the neighboring C.W. Mining operations. To calculate travel distances, one needs to divide the travel path into segments, each having specific frictional and rebound characteristics. To get a range of possible travel distances, the program simulates movements of a number of rock blocks and creates a histogram of travel distances, velocities, and rebound heights at any specific analysis point. The size of a rock block is an important parameter and can be easily estimated by using either joint spacing at the source or by measuring the maximum size of rock that has traveled down failed slopes.

6.2 Simulated Rock Movements

Figures 6 and 7 present the results for west Bear Canyon and Wild Horse Ridge locations A-A' and B-B', respectively. Attachment D includes additional information regarding travel distances, block velocity, and rebound heights. These simulations are based on 25 spherical rock blocks 6 ft in diameter. This diameter was selected conservatively, considering both the actual spacing of joints at the source and observed sizes of rolled blocks along the west Bear Canyon location (.5 to 6 ft).

Figure 6 is in general agreement with observations of travel distances, considering the accuracy of topographic data near section A-A'. This section consists of nine mostly gently dipping segments, and thus the calculated travel distances are limited to 770 ft from the source (Castlegate Sandstone). The estimated travel distances are larger for steeper canyon slopes at the location of B-B', which can reach a maximum distance of 1,200 ft from the base of the Castlegate Sandstone. Figure 7 presents rock travel trajectories along the 13 segments forming this profile.

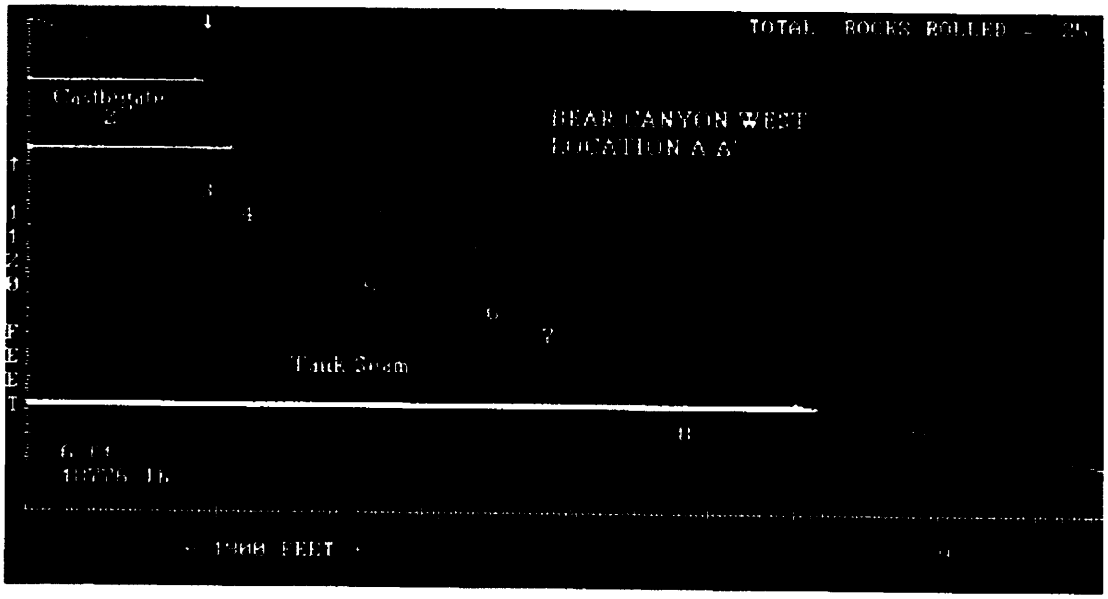

Figure 6. Simulated rock travel distances and trajectories for section A-A' (figure 2).

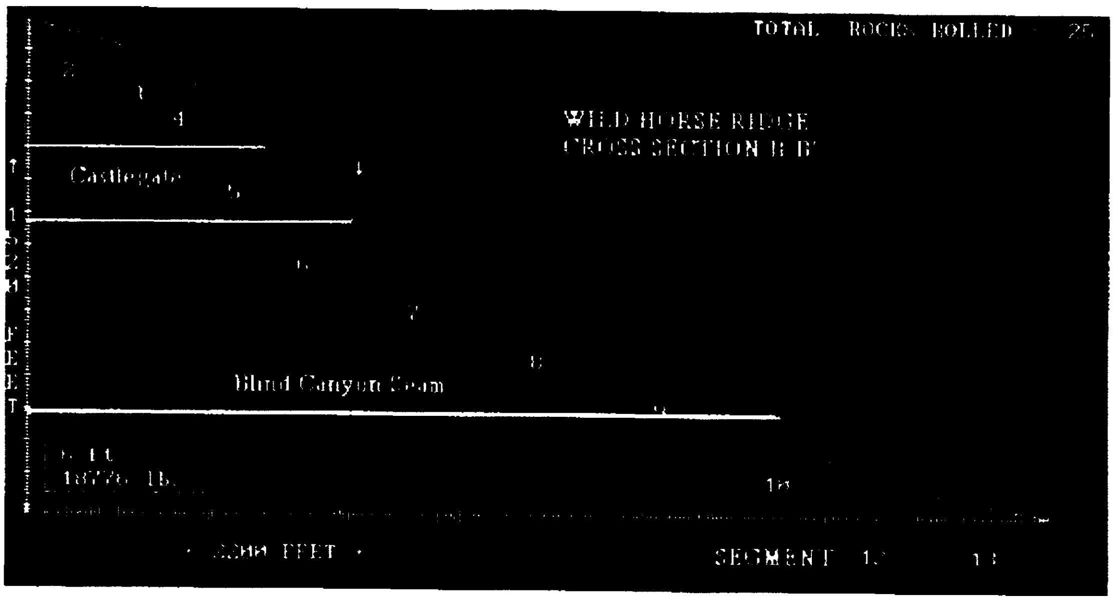

Figure 7. Simulated rock travel distances and trajectories for section B-B' (figure 2).

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ATTACHMENT A

COPIES OF PERTINENT PUBLICATIONS

DEVELOPMENT OF A STATISTICAL TECHNIQUE FOR ASSESSING SANDSTONE ESCARPMENT STABILITY

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ABSTRACT

During the last decade, a significant amount of research has been conducted by Energy West Mining Company, government agencies, academia, and consulting companies to develop predictive tools for assessing the stability of the Castlegate Sandstone, which is found approximately 250 m above multiple-seam coal reserves. Energy West Mining uses longwall mining techniques in its operations near Huntington, Utah, and these studies were initiated to satisfy requirements for maintaining the stability of the Castlegate Sandstone and resource recovery.

In this study, the authors have used multiple-regression analysis techniques and a wealth of data collected over many years on geology, mining, and escarpment stability. The volume of failed rocks is used as the response variable after several other factors were considered, including measurement of surface deformation and frequency of mining-induced surface fractures. Geologic and geometric variables were obtained along 3.7 km of escarpment exposure at 130 study locations. Regression analysis of data for the first 29 study locations, which had been fully undermined, showed that surface topography played a critical role in influencing escarpment stability. Preliminary regression analysis results from 70 study locations identified several other important geologic and mining factors that influence the stability of the sandstone escarpment. These are canyon slope, sandstone thickness, and mining influence angle.

INTRODUCTION

This paper presents progress being made in developing a predictive statistical model as a tool for assessing the stability of escarpments in the vicinity of Energy West's longwall operations near Huntington, Utah. Such models are ideal for probabilistic risk analysis so that the economic benefits of extracting coal reserves can be compared to the likelihood of escarpment instability.

There are two methods routinely used by engineers and researchers to help predict what conditions will be in the future: statistical and computational. Starfield and Cundall (1) identify rock mechanics problems as "data-limited," that is, one seldom knows enough about a rock mass to use computational methods unambiguously. These methods, however, are extremely useful for studying failure mechanisms and testing different hypotheses about the cause of failure. Statistical methods, on the other hand, are uniquely capable of being applied where there are good data, but a limited understanding of certain phenomena, such as the mechanism of escarpment failure (toppling, pure translation, or a combination of these and other mechanisms).

Various investigators from both the U.S. government and universities have used computational techniques for analyzing surface subsidence and escarpment failure mechanisms. The results are in general agreement with studies in the Sydney Basin of Australia (2). A combination of two-dimensional, boundary-element (3), finite-element (4) and discrete-element formulations was used in the U.S. studies. To overcome the limitations of using small-strain, continuum finite-element methods, a hybrid approach was used. In this approach, finite-element deformation was imposed on a detailed discrete-element model of the escarpment and the mudstone foundation and incorporated both horizontal slip planes and vertical joints (5). Researchers from the U.S. Bureau of Mines (6) also completed a few preliminary three-dimensional, finite-element modeling studies. While successful in analyzing failure patterns and mechanisms, these studies have clearly identified the limitations of numerical modeling techniques in matching measured surface deformation because of the data-limited nature of these modeling efforts.

Statistical and semi-analytical techniques have been used alternatively for many rock mechanics problems where there are good data but limited understanding of some natural phenomena, such as rock bursts (7), creep (8), and ground support (9-10). Australian researchers (11) have used the results of comprehensive field investigations with other data analysis techniques to identify the influence of individual factors (such as horizontal movements and cliff heights) on cliff stability. Multivariate statistical evaluations of these results are awaiting additional investigation.

The technical approach in this study consisted of incorporating and digitizing data on geology, mining, and escarpment stability collected over many years in several mining areas into a statistical model. This model is being used by mine personnel for routine assessments of escarpment stability in new mining areas even as new data are being incorporated to enhance model predictions. Model input consists of geologic and mining conditions, including escarpment geometry, orientation of joints with respect to the escarpment, joint density, joint continuity, and mining influence angle.

The authors implemented the first phase of the study during 1997, collecting detailed geologic and mining factors at 29 study locations, each 30 m wide. Phase 1 consisted of (1) characterization of geologic, mining, and response variables in the study area and (2) analysis of patterns seen in the data and identification of important variables through the use of multilinear regression models. In the Corncob Wash study area, which is the primary focus of this paper (figure 1), the 1,000-m-long escarpment exposure provided the opportunity to observe surface effects and evaluate factors that contributed to escarpment instability after mining had been completed. In the Rilda Canyon study area, pre-mining conditions were characterized in detail, and postmining conditions will be observed in the near future as both the Blind Canyon and the Hiawatha seams are mined.

During the second phase of the project, post-mining conditions at 41 study locations in the South Newberry study area were characterized along the 1,200-m-long escarpment. Additional analyses are planned when mining in the Rilda Canyon is complete (phase 3).

CHARACTERIZATION OF GEOLOGIC, MINING, AND RESPONSE VARIABLES

The first step in developing predictive statistical models was to create suitable numerical values that expressed geologic and mining conditions in the study area (figure 1). The second step was to reduce the number of independent variables by combining some existing variables into new categories and identifying highly correlated independent variables. Reducing the number of variables is needed when there are too many variables to relate to the number of data points. The presence of highly correlatable variables influ-

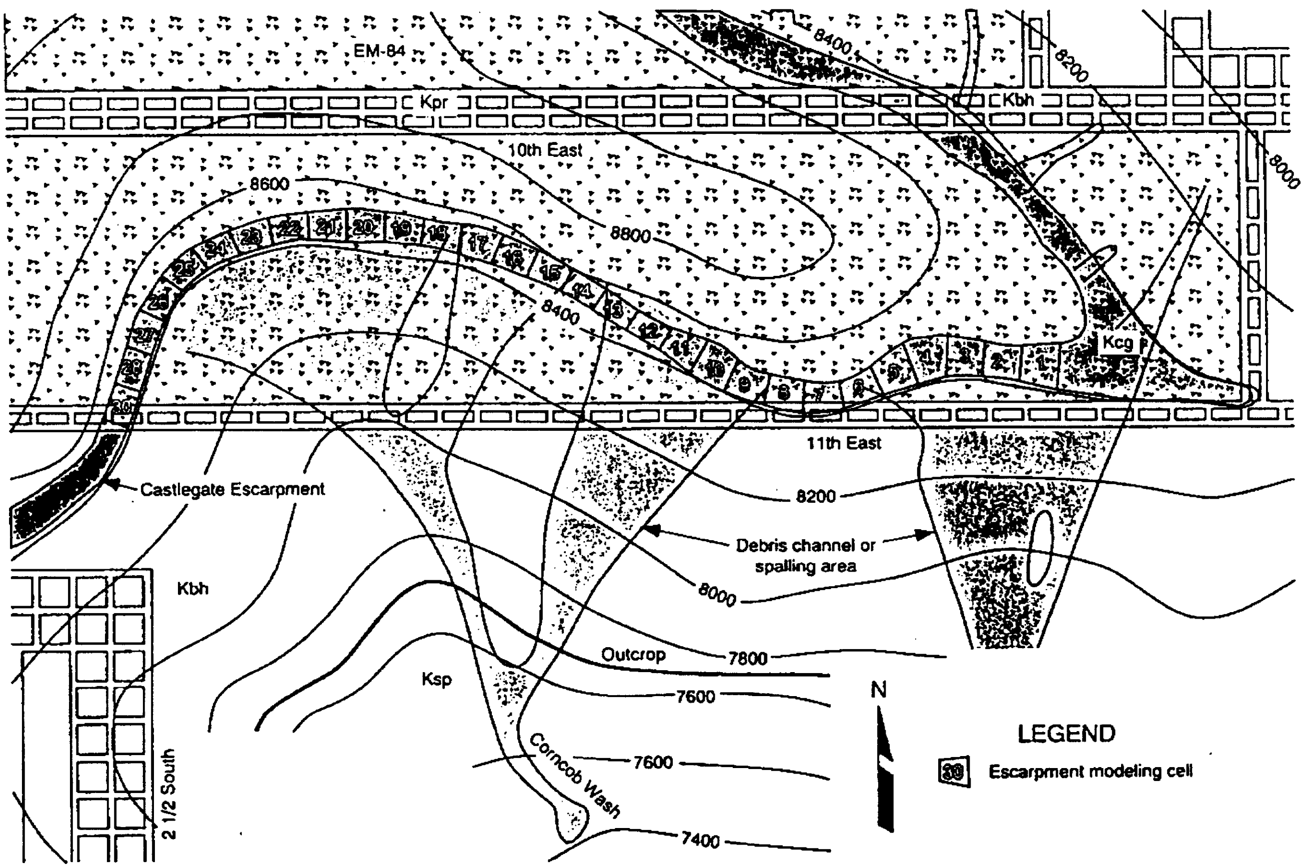

Figure 1. Corncob Wash study area

ences what procedures are selected for multiple-regression analyses. The third step was to develop a multivariate regression model and identify significant factors that contribute to escarpment stability.

The study areas were partitioned into cells approximately 30 m wide. This resulted in 29 cells for the Corncob Wash study area and 41 cells in the South Newberry area. The authors estimated geologic, mining, and response variables for individual cells based on field mapping, examination of borehole logs, and aerial photographs obtained before and after mining. Table 1 presents statistical information, including mean and standard deviation for the first 29 cells in the Corncob Wash study area.

Most variables are self explanatory. A brief description of some of the variables (in italics) is given below.

- *Joint sets 1 and 2* are the primary and secondary persistent joint sets mapped in each area.

- The *angle between joint sets and an escarpment* can influence escarpment stability, a hypothesis that is based on observations of subsidence-related fracturing in the western United States (12). Using this hypothesis, an escarpment may have a higher probability of failure where the angle between joints and the escarpment (or mining boundaries) is small (0° to 30°).

- The *excavation width-to-depth ratio* is similar to a subsidence engineering term (13) that relates the total width of an excavation to the average depth of cover over the panel of interest. This ratio measures changes in subsidence mode as excavations are widened during mining of successive panels. As the ratio approaches 1.4, a supercritical subsidence stage is reached.

- Based on a review of mining maps and experience in Newberry Canyon (4-5), *escarpment shape* (convex or concave) appears to influence escarpment stability and thus is included as a geologic variable. Observations in Newberry Canyon by researchers from the University of Utah indicate that virtually all of the failures occurred in a concave portion of the escarpment. A hypothesis was that natural erosion of the escarpment took place at a faster rate at these locations as a result of greater premining joint density (5).

Table 1. Population statistics for the Corncob Wash study area

Variable	Mean	Standard deviation
A. Geologic:		
Angle between joint set 1* and escarpment, deg	27	27
Angle between joint set 1 and longwall face, deg	61	24
Angle between joint set 2* and escarpment, deg	57	26
Angle between joint set 2 and longwall face, deg	19	12
Joint set 1 spacing, m	9.5	12
Joint set 2 spacing, m	9.5	14
Horizontal continuity for joint set 1, m	6.4	6.5
Horizontal continuity for joint set 2, m	8.5	7.6
Vertical continuity for joint set 1, m	4	3.6
Vertical continuity for joint set 2, m	2.4	0.8
Joint set 1 and escarpment index	1.1	1.7
Joint set 1 and longwall index	1.2	1.8
Joint set 2 and escarpment index	0.2	0.8
Joint set 2 and longwall index	0.4	1.2
Joint set 1 index	3.9	5.2
Joint set 2 index	0.9	1.0
Erosion under escarpment index	0.1	0.3
Escarpment shape index	0.9	0.3
Canyon slope, percent	80	9
Escarpment slope, percent	218	45
Thickness of Castlegate Sandstone, m	76	6.4
Seam-to-sandstone distance, m	237.8	5.5
B. Mining:		
Influence angle, deg	71	11
Excavation width-to-depth ratio	2.5	0.6
C. Response:		
Failure index	1.4	0.8

- The *influence angle* is defined as the angle from a horizontal plane and a line from the mining limit to the base of the Castlegate escarpment (figure 2). This angle is 90° where the escarpment is directly above the mining limit and over 90° in areas outside the mining limit.

Several indexes were created to combine joint data from various data sets into a single variable

- The *joint set I and escarpment* index (or INJSiE) took values 0 to 4, depending on the amount of deviation between a joint set and the escarpment (figure 3).

- The *joint set I and face* index (or INJSiF) took values 0 to 4, depending on the amount of deviation between a joint set and a longwall face.

- The *joint set I* index is a cumulative measure of joint orientation and block size —

  INJSiE + INJSiF + horizontal continuity × vertical continuity ÷ spacing^2 where INJSiE = the joint set I and escarpment index and INJSiF = the joint set I and face index.

- The *erosion under escarpment* index equals values of 1 and 0, depending whether the area under the escarpment at the particular cell is eroded or not.

- The *escarpment shape* index equals values of 1 and 0 for concave and convex escarpment geometries.

- The *failure* index equals values of 0, 1, and 2, depending on the estimated volume of failed material within the cell of interest. The failure index was selected from among other response variables, including tensile cracking and vertical and horizontal movement on the surface, because it best describes the stability of the escarpment and can be estimated for each cell. The failure index is used as a dependent variable in regression analyses.

TRENDS IN DATA AND IMPORTANT VARIABLES

Because there are many variables that could influence the stability of the escarpment (table 1), it is important to study trends in the data and use prudent statistical procedures that take into account the interrelationships among independent variables. To study these relationships, a bivariate correlation matrix was constructed to measure the linear correlation among geologic, mining, and response variables. The correlation matrix includes correlation coefficient, number of data points, and two-tailed significance tests. The correlation coefficient (r) indicates the strength of linear relationships between any pair of variables.

Based on a review of the correlation matrix, the authors found fair correlation between the failure index variable and several independent variables, as well as among some independent variables. For example, the correlation coefficients between the failure index variable and the escarpment shape and influence angle variable are 0.58 and -0.48, respectively. However, escarpment shape and influence angle happen to have fair correlation as well (correlation coefficient equals -0.48). Thus, there is an interrelationship among the independent variables that can be taken into account using step-wise inclusion of these independent variables while conducting multiple-regression analyses (7).

To identify important factors that contribute to escarpment stability, a multiple-regression analysis was used. Escarpment stability was estimated using the failure index as the dependent variable. The multilinear regression procedure consisted of entering independent variables one at a time into the equation using a forward selection method (14). In this method, a variable is entered into the equation using the largest correlation with the dependant variable. If a variable fails to meet entry requirements, it is not included in the equation. If the first variable meets the criteria, the second variable with the highest partial correlation is

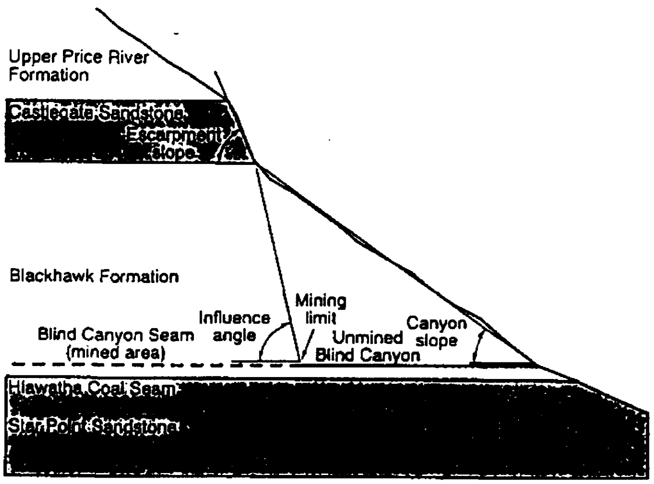

Figure 2. Mining and escarpment geometry

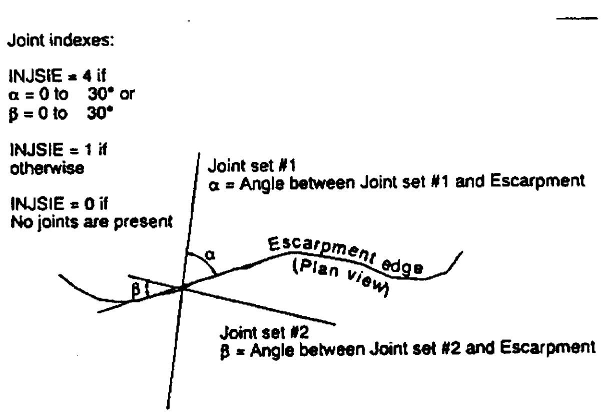

Figure 3. Escarpment geometry and geological discontinuities

then selected and tested for entering into the equation. This procedure is very good when there are hidden relationships among the variables. The multiple correlation coefficient, R, which is a measure of goodness-of-fit, for the last step is 0.68.

Based on an examination of standardized regression coefficients for the first 29 cells, the following variables best explain variations in the failure index.

- *Escarpment shape* index. The lower the escarpment shape index, the smaller the failure index. This is in agreement with experience in Corncob Wash and other areas where convex areas have historically remained stable when undermined.

- *Joint set 2 and escarpment* index. This is the only geologic variable that contributes to goodness-of-fit in a mathematical sense. Because there are very few secondary joints mapped in the Corncob Wash study area, it is not clear to the authors how relevant this factor is to escapement stability.

New data from the South Newberry Canyon have recently become available and were incorporated in the model during phase 2. Preliminary results using a total of 70 cells have identified several other important mining and geologic factors, including *canyon slope*, *thickness of Castlegate Sandstone*, and *mining influence angle*. The failure index is higher at locations where the canyon slope is steeper, the Castlegate escarpment is thicker, and full extraction is completed beneath the escarpments.

CONCLUSIONS

Based on the regression model and observations of escarpment behavior prior and after longwall mining, the authors have identified important factors influencing escarpment stability. It is shown that surface topography (escarpment shape) played a critical role in influencing escarpment stability and that convex escarpment geometries remained stable after the area had been undermined. Work is in progress to enhance existing models by including data from other study areas that have different geologic, geometric, and stability conditions. Preliminary results from a new study area (South Newberry) have identified several other important factors, including canyon slope, thickness of the Castlegate Sandstone, and mining influence angle.

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20th International Conference on Ground Control in Mining

FIELD EVALUATION OF MOBILE ROOF SUPPORT TECHNOLOGIES

FINAL

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Abstract

This study presents a historic overview of the role of mobile roof support (MRS) technologies in improving stability and worker safety and presents the results of recent field evaluations of the MRS load rate monitoring device and other remote deformation-monitoring techniques. Field studies were implemented at two sites in cooperation among researchers from the National Institute for Occupational Safety and Health (NIOSH), Maleki Technologies, Inc., and J. H. Fletcher & Co. The objective of the field programs were to (1) study the interaction between MRS's and coal mine strata and (2) develop and test suitable monitoring systems for assessing roof and pillar stability. An MRS consists of a roof canopy, four hydraulic cylinders, a caving shield canopy, and associated electromechanical systems mounted on crawler tracks. The machines are controlled by radio from a remote location and operate on self-contained power units. Typically, MRS's have capacities of 5,340 and 7,120 kN (600 and 800 tons). In comparison to posts, an MRS is capable of maintaining the yield load after significant amounts of roof-floor deformation. Because the mining cycle is accelerated, MRS's help reduce the potential for time-dependent roof falls.

MRS performance has been monitored in the laboratory under controlled static loading conditions and in the field under deep, two-seam mining conditions. Laboratory studies have quantified support capacity and system stiffness as a function of machine height. Field investigations have focused on determination of optimum operating conditions and development of warning systems that indicate excessive load on the machine and/or impending roof-pillar stability problems. Analyses of field data show that roof instabilities are influenced by (1) pillar failure, (2) pillar yielding, (3) mine seismicity, (4) geologic structures, and (5) panel layout designs and mining practice. Pillar yielding and failure (unloading) and seismicity can be conveniently monitored by the load rate monitoring device, but for consistent detection of roof falls, additional deformation measurements directly within the cuts are needed.

INTRODUCTION

Room-and-pillar mining is one of the oldest methods used for the extraction of tabular ore bodies. In this method, a series of rooms are driven on advance using continuous miners and shuttle cars while the roof is bolted a short distance behind the face. During the retreat, the same equipment is used to mine the pillars, which allows roof rocks to cave behind the face. To control the cave line, a series of secondary support systems are installed as mining continues within the pillars.

The room-and-pillar mining method is at a disadvantage when compared to other mining techniques, such as longwall mining. Because of economies of scale, productivity using room-and-pillar mining is significantly lower. The longwall method is also much safer because the retreat is completed under the protection of self-advancing hydraulic support systems at the face. However, during the last two decades, federal laboratories, mining companies, equipment manufacturers, and geomechanics consultants have cooperated to improve the understanding of strata mechanics and develop a remotely controlled, self-advancing support system called a mobile roof support (MRS). This cooperation has resulted in improvements in the safety and productivity of room-and-pillar retreat operations.

Figures 1 and 2 illustrate generic panel layouts and pillar extraction sequences for two typical room-and-pillar retreat systems. The first is three-entry access and retreat to one side, while the second is nine-entry access with full retreat within the panel. In the first system, mining starts by driving a three-entry panel access to the boundaries of the room-and-pillar panel. A three-entry system using narrow rib pillars is developed to the side and retreated. After pulling one row of pillars, another row is driven into the solid coal block, and the sequence is repeated until the panel coal is extracted. Pillar recovery operations consist of splitting the pillars and fenders. Figure 1*A* presents the mine layout at four stages of pillar recovery. Figure 1*B* shows the sequence of the pillar cuts, typical position of posts, and the location of unmined stumps for the extraction of a pillar using the split-and-fender method.

In the second system, a nine-entry access is developed on advance to the panel boundaries. The pillars are then extracted until the entire panel is mined. Figure 2*A* presents a panel layout and the location of MRS's at three intermediate stages of pillar recovery using the "Christmas tree" method. Figure 2*A* shows the sequence of cuts taken from two pillars where MRS's are used as secondary support. Many variations in these two panel layouts and excavation sequences are

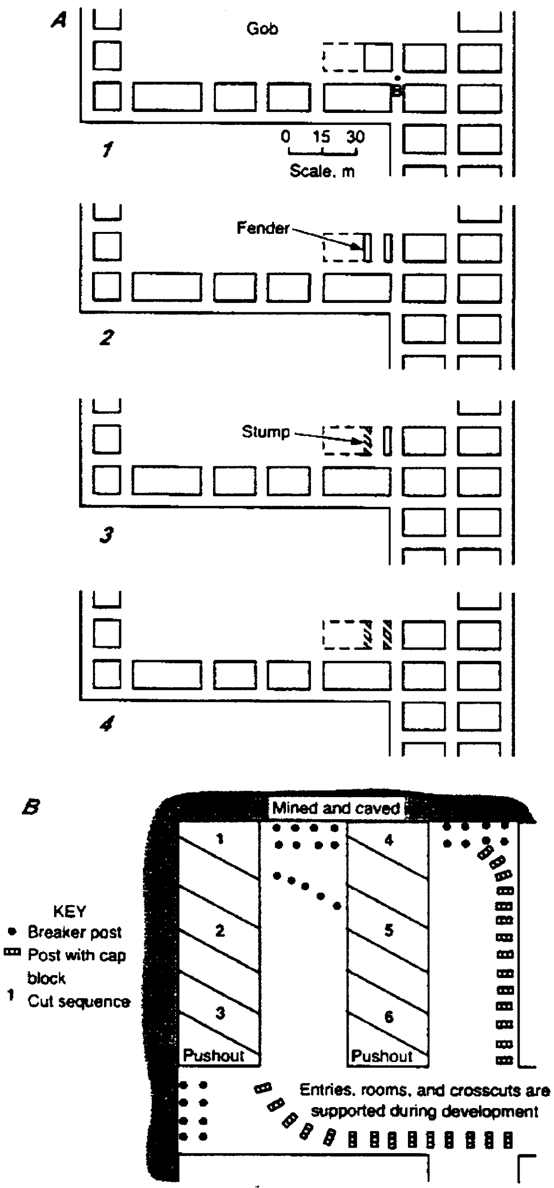

Figure 1.—Mine layout (*A*) and pillar extraction sequence (*B*) using split-and-fender method with posts.

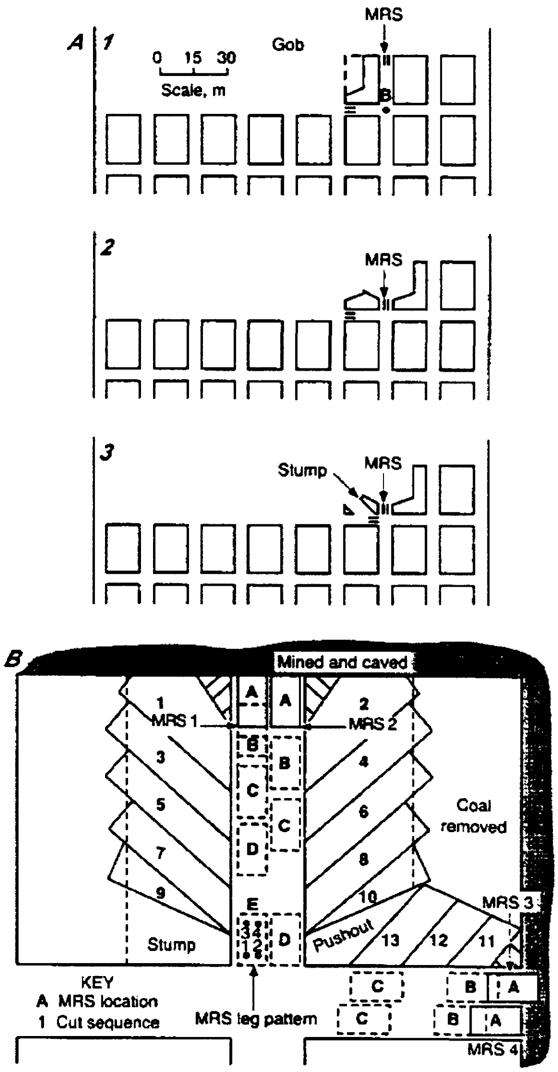

Figure 2.—Mine layout (*A*) and pillar extraction sequence (*B*) using Christmas tree method with MRS's as support.

practiced in U.S. coal mines. New applications of the three-entry system involve use of MRS's instead of posts and eliminates fenders completely.

After completing an analysis of the hazards of room-and-pillar retreat mining systems, it became apparent to the authors that safety could be significantly improved by considerations of (1) human factors, (2) remotely controlled MRS's, (3) mine layout designs, and (4) ground monitoring systems. A significant effort was directed to studying the above factors both in the laboratory and in the field. Recent geomechanics field evaluations focused on identifying failure mechanisms and critical levels of load and movement rates that are indicative of impending stability problems.

HUMAN FACTORS

Several human factors considerations were identified during an earlier geomechanics field study (Maleki 1981) in which the main objective was to identify causes of roof stability problems and develop practical monitoring techniques for detecting these problems (Maleki and McVey 1988). These factors were (1) the number of people required at the face, (2) the amount of time required to work at the cave line, (3) poor footing in entries, which influenced timely escape during a roof fall, (4) worker reaction at the time of a roof fall, and (5) the judgment-based methods used by miners to evaluate the stability of the roof and determine the optimum time for retrieving miners and equipment.

A large crew is required for conventional room-and-pillar retreat operations because posts must be delivered, cut to size, and installed. Each installation takes approximately 20 minutes and requires two to three workers. Debris on a mine floor can accumulate quickly and create poor footing. Miners must judge roof stability continually on the basis of observations of primary and secondary support behavior (bending of roof plates, crushing of posts, etc.). In the study mine, when the roof caved prematurely and trapped a miner in a cab, other miners rushed to help. A second rock fall could have resulted in serious injury to rescuers (Maleki 1981). This could have easily happened, considering that many posts had already been broken and some had been knocked down during the first fall.

DEVELOPMENT AND TESTING OF MRS

To improve the safety of room-and-pillar retreat systems, a two-step solution was proposed. First, the mechanics of strata behavior was studied through extensive field measurements, and practical techniques for assessing roof behavior were developed. Second, a prototype of a remotely controlled roof support system was developed to eliminate the need to install posts near the gob. The machine was equipped with a dozer blade so that floor debris could be cleaned routinely, which allowed easier travel and escape. The prototype unit was developed by the U.S. Bureau of Mines in cooperation with an equipment manufacturer and a mining company (Thompson and Frederick 1986).

Commercial units have since been developed by U.S. and Austrian manufacturers and are being used on two continents. The commercial MRS units are more rugged and have higher capacities (5,340 to 7,120 kN [600 to 800 tons]) (Wilson 1991; Howe 1998) than the prototype. They consist of a roof canopy, four hydraulic cylinders, a caving shield canopy, and associated electro-mechanical systems mounted on crawler tracks. The system has radio control and self-contained power units. Because of their greater mobility and because they allow higher resource recovery, they are currently being used in 36 U.S. coal mines, as well as a number of Australian mines (Shepherd and Lewandowski 1992; Habenicht 1988).

MRS performance has been monitored both in the laboratory and in the field by NIOSH and MTI personnel. Laboratory investigations focused on an evaluation of support stiffness and load-carrying capacity under controlled static loading conditions. The study quantified system stiffness as a function of machine height for both two- and three-stage hydraulic cylinders (Barczak and Gearhart 1997, 1998). The advantage of the three-stage cylinder design is greater operating range, but a disadvantage is reduced support stiffness. Each unit has the load-bearing capacity of six posts and the stiffness of two hardwood posts (Barczak and Gearhart 1997). The study also identified inaccuracies in hydraulic cylinder pressure measurements of roof loads when the bottom cylinder stages were fully extended.

The mechanics of load transfer from pairs of MRS's to the mine strata was analyzed using laboratory results, boundary-element modeling, and analytical solutions. The results showed that MRS's support roof rocks near the machines, but do not have the capacity to control overall roof-floor convergence and overall stress distributions because the MRS's are considerably less stiff than coal-measure rocks. In comparison to posts, however, an MRS is capable of maintaining the yield load after significant amounts of roof-floor deformation. Because the mining cycle is accelerated, MRS's help reduce the potential for time-dependent roof falls.

To study the influence of pairs of MRS's on the mine roof, the authors used analytical solutions for two pairs of MRS's positioned 5.5 m (18 ft) apart (figure 3) (Maleki and Owens 1998). Results showed that MRS's form a pressure arch in the immediate roof that reduces the potential for roof falls in the space confined by the MRS's. This is beneficial for protecting a continuous miner when it is operating within this space. It was also found that higher MRS capacities and setting pressures are useful for stabilizing the upper strata, but may contribute to differential loading on the immediate roof, failure of mechanical bolts, and reduction in the stability of the immediate roof.

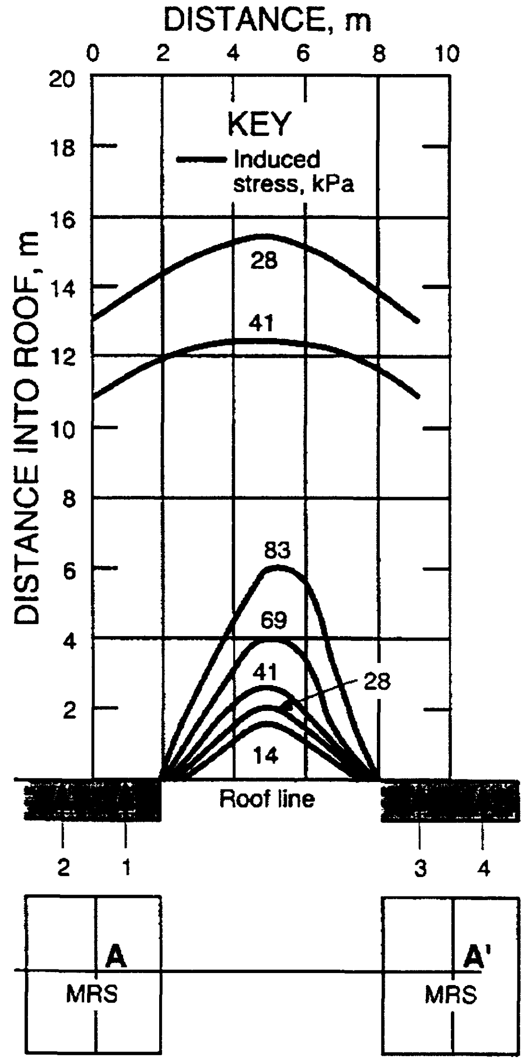

Figure 3.—Stress isobars along A-A' for two pairs of MRS's at 5.5-m spacings.

Early field evaluations focused on a comparison of ground movements in two room-and-pillar retreat sections using the split-and-fender method with posts (figure 1) and the Christmas tree method with MRS's as the secondary support system (figure 2). In addition, the history of hydraulic pressure was analyzed for all four MRS legs (Hay et al. 1995). Deformation measurements indicated generally higher strata movement at the intersections in the section using the Christmas tree method. Because of differences in geologic conditions and mining practices, it was not possible to make a direct comparison. We recommended that numerical modeling of these geometries address mine layout designs while keeping geologic conditions constant.

PANEL LAYOUT DESIGN

Early field studies identified the importance of mine layout designs and revealed the dangers of overconfidence concerning the ability of MRS's to support the entire area. Such overconfidence contributed to workers choosing unsafe operating locations. Thus it became apparent to the authors that to improve stability, layout designs that control convergence and stress should be developed. To illustrate this point, boundary-element analyses were completed in which stress distributions were calculated in both single and multiple seams. These analyses were also helpful in tailoring the type of monitoring required to assess changes in the stability of the mining system.

The first study compared stress distribution and convergence patterns for two pillar recovery plans: split-and-fender and Christmas tree. Model input was based on extensive laboratory and field measurements in one mine (Maleki 1981), and modeling procedures were based on a methodology developed for coal mine excavations (Maleki 1990; Maleki and Owens 1998). The analyses were completed for a typical depth of 305 m (1,000 ft).

Figure 4 presents the calculated roof-floor convergence for a point in the intersection for two pillar recovery methods (point B in figures 1A and 2A) and provides guidance for selecting monitoring systems. Note that calculated deformation significantly increases within a mining step, which is associated with the failure of fenders and stumps. MRS's will therefore experience an increase in both vertical and lateral support loading as fenders fail. Since fender failure induces differential movement in the mine roof, a roof fall may be triggered. Such a roof fall may be sensed through monitoring either roof movements or possibly MRS leg pressures. The change in convergence that occurs as a result of failure of the fenders is large enough to cause a change in leg pressure and can be conveniently detected by the load rate device. Obviously, changes in convergence and roof movements may best be directly detected by monitoring roof-floor movements (Maleki 1981) if inadequacies in measuring the hydraulic leg pressures of the MRS's are suspected.

Roof-floor convergence is at least 10% higher using the Christmas tree method, as illustrated in figure 4. To control convergence, a stump is left in the model (figure 2). Further improvements in stability and convergence can be achieved by changing the size of the stumps and pillars left behind while considering site-specific structural conditions (i.e., using engineered mine layouts and extraction designs).

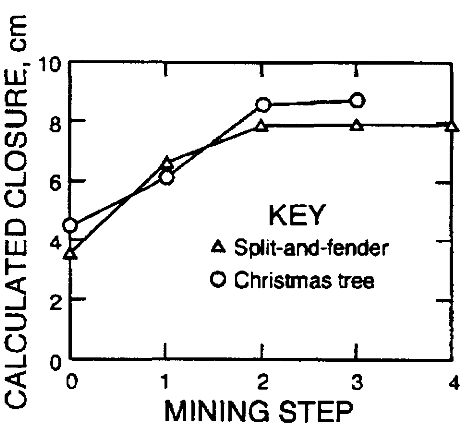

Figure 4.—Calculated closure for split-and-fender and Christmas tree methods at location B.

MRS's are used often when mining difficult reserves, such as where there are earlier workings in adjacent seams. In a second study in a cooperating mine described here, numerical modeling techniques and field data were used to show how two-seam layouts influence stability and support response. The mine uses the room-and-pillar technique to extract three-seam reserves in coal fields on the Wasatch Plateau near Huntington, Utah. These seams are located toward the base of the Blackhawk Formation and consist of the Tank, Blind Canyon, and Hiawatha. The Tank Seam is presently being mined in an area partially undermined by the Blind Canyon Seam, approximately 85 m (280 ft) below. Thus, mining layout and pillar pulling plans are complex. The test site in the Tank Seam is located in a graben bordered to the east and west by two major faults (figure 5). North-south-trending joints are common in the section and influenced by mining and caving process during extraction of the Blind Canyon Seam.

Figure 6 presents Tank Seam mining geometry and vertical stress distributions over a portion of the two-seam mining areas during the extraction of pillar 2. A stress profile was also prepared (figure 7) along an east-west cross section positioned toward the middle of the modeled area (away from the active face in the 1st North Mains). Modeled geometry in the Blind Canyon Seam was limited to the fully retreated 2nd East panel. This panel lies directly under the 1st Main North, but shifts some 24 m (80 ft) toward the east and so the last (most westerly) row of 1st North Mains pillars is not undermined. Modeled areas in the Tank Seam include a fully retreated room-and-pillar panel to the western boundary of the model (top of the page) and a 43-m- (140-ft-) wide barrier pillar between this gob and the 1st North Mains (figure 5)

Results indicate a nonuniform stress distribution over the 1st North Mains. Pillar stresses increase to the west across the 1st North Mains. Maximum stresses are concentrated over the last row of pillars (row 8) and the barrier pillar. This is in agreement with underground observations indicating large amounts of rib spalling and floor heave near pillar 8 in contrast with little (unnoticeable) movement to the east. This two-seam mining geometry created an opportunity to assess machine performance in the field under these two different loading conditions.

Based on this and other multi step stress analyses, it became apparent that pillar stresses exceeded pillar strength [21 MPa (3,000 psi)] (Maleki 1992) when approximately half a pillar was extracted. At this time, the pillar exceeded yield loads and was approaching the post-failure regime. Seismicity noticeably increased. Pillar unloading resulted in increased roof-floor convergence and load transfer to the MRS units nearby. This process was associated with an increase in the rate of loading on the MRS's. Roof falls may be triggered by additional movement, particularly if smooth joints are present.

DEVELOPMENT OF GROUND MONITORING SYSTEMS

During field tests in underground mines, the authors identified three factors that might adversely influence worker safety in an MRS section.

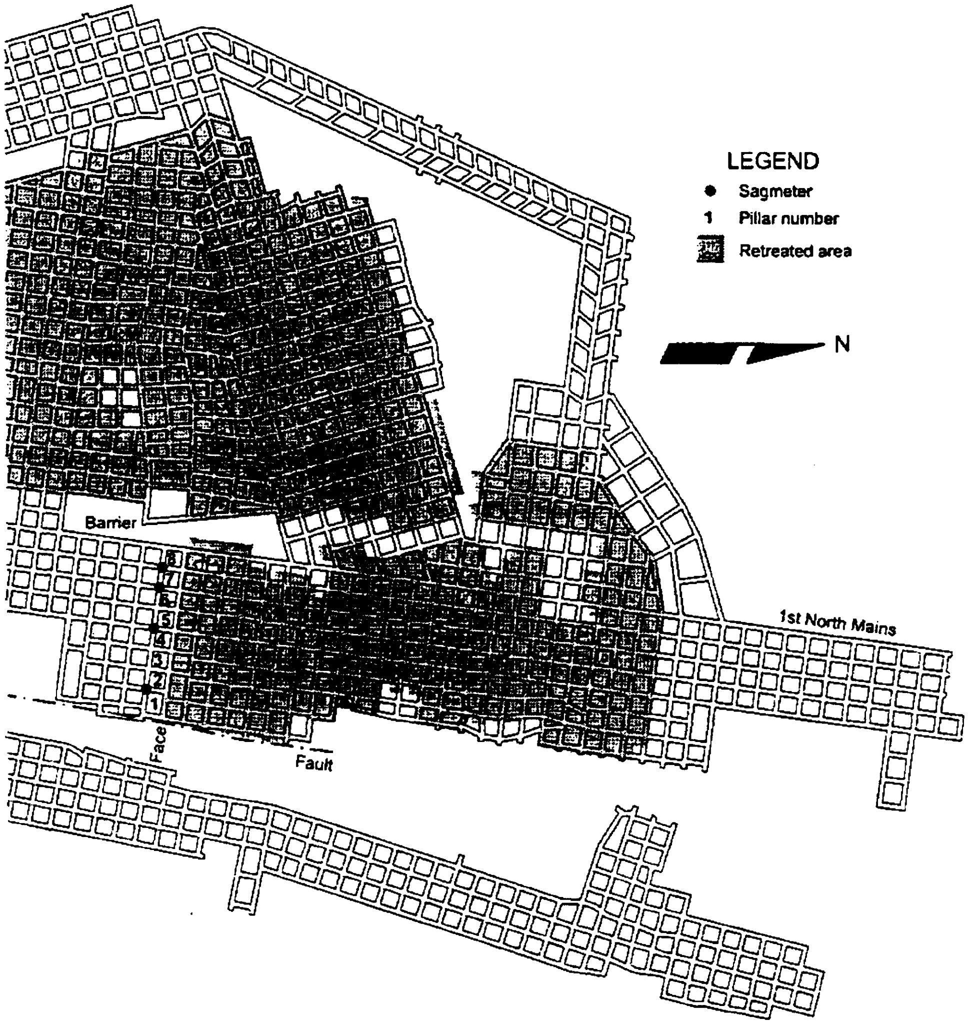

Figure 5.—Mining geometry and monitoring locations in Tank Seam.

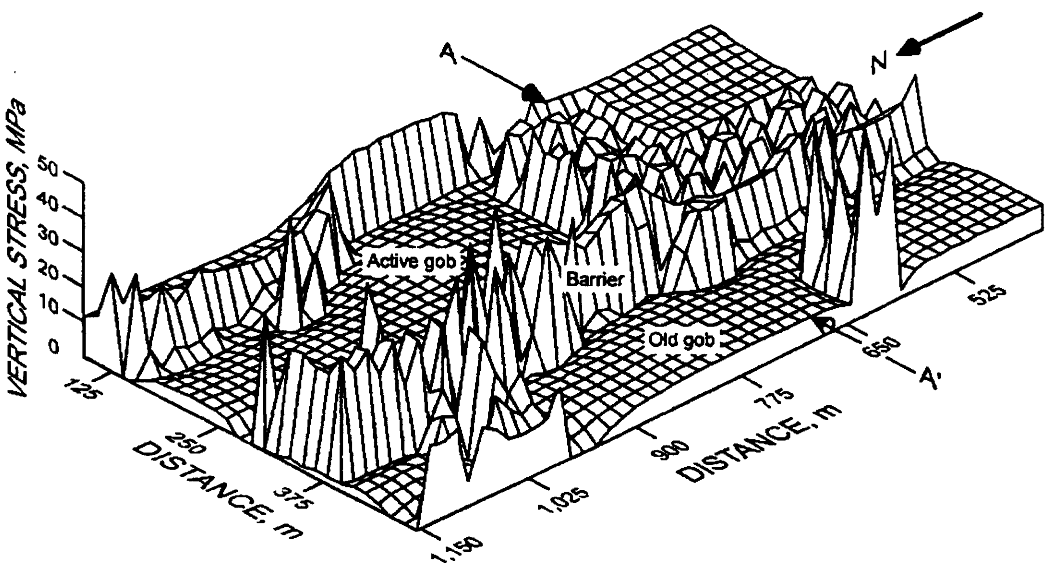

Figure 6.—Vertical stress distribution on the Tank Seam.

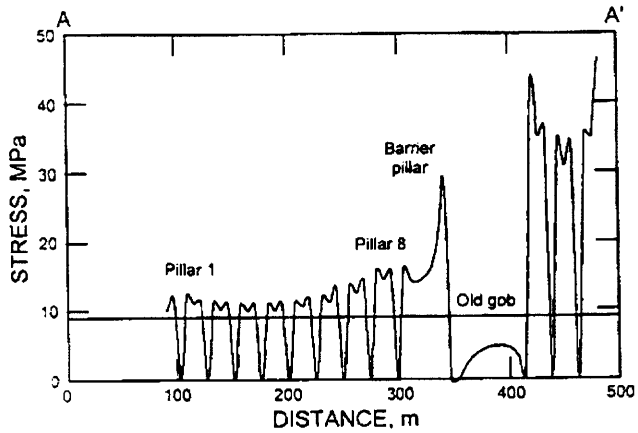

Figure 7.—Vertical stress profile along section A-A'.

- Elimination of posts reduced a worker's ability to assess roof conditions.
- Overconfidence in the ability of MRS's to support the entire area caused some miners to chose unsafe operating positions.
- Use of MRS's on a routine basis under adverse geologic and mining conditions to recover reserves that were otherwise unminable.

It became apparent to the authors that there was a need to develop a warning system that would alert workers to unstable roof conditions so that miners and equipment could be removed before a roof fall occurred. Two monitoring methods were chosen on the basis of mine measurements and numerical modeling considerations. These were load-rate monitoring on the hydraulic legs of MRS's and remote monitoring of roof movements using a theodolite. A reliable warning system needs to combine both ground deformation and load-rate data. Theodolite and spads have been effectively used for the remote measurement of roof movements and for the detection of roof falls in room-and-pillar operations (Malcki 1981). In this application, marked spads are installed in the area of interest during pillar recovery and ground movements are remotely monitored using a theodolite (or transit). By measuring the change in vertical angle, the rate of roof movement is calculated.

A load rate monitoring device was developed by NIOSH that monitors dynamic loading rates on an MRS in real time and displays warning signals. Hydraulic supports such as the MRS provide little or no discernible audible or visual indications of impending roof stability problems. In MRS retreat mining sections, miners rely on the hydraulic gauges on the MRS's to determine when to cease operations and leave the area of the active mining face before a roof fall. An imminent roof failure is sometimes preceded by a rapid increase in pressure on the dial gauges. However, these gauges are difficult to read, requiring miners to approach the MRS to monitor the gauges, which in turn requires them to be close to the active mining face, an area susceptible to roof falls, and in a location with a lot of equipment activity. As a result, miners do not check the pressure gauges often. Monitoring the rate of loading on MRS legs was shown to provide warnings about major events, such as failures of fenders and pillars. These events often trigger roof falls.

With the cooperation of the MRS manufacturer, J. H. Fletcher & Co., the device was installed and tested on MRS's in the laboratory and in the field. The system is MSHA permissible and operates as an integral part of the MRS. Research continues in analyzing the data from recent field installation and in identifying critical loading parameters associated with roof and/or pillar stability problems. Necessary calibration can be done prior to installation or periodically as mine conditions change, but need not be done by operating personnel at the mine. The operating parameters for the system are set by connecting the system to a laptop computer via an RS-232 null modem cable with the communication terminal emulator acting as the laptop client program. This allows a trained user to change the parameters for triggering the various load rate indicator devices easily to suit conditions at the mine.

FAILURE MECHANISMS AND MONITORING RESULTS

MRS performance was monitored during the extraction of one row of pillars. Hydraulic leg pressures were collected on all four MRS's using Campbell Scientific[1] data acquisition systems. Two loading rates are used to analyze the loading history of MRS's: (1) Instantaneous loading rate calculated by taking measurements every 2 sec and (2) average loading rates calculated by taking measurements following an acceleration in loading rates up to a period of 1 hour. The instantaneous rate is highly variable but useful when addressing seismically induced events. The average loading rate is more suitable for addressing overall changes in pillar stability.

Three time windows were selected to analyze failure mechanisms and machine response to applied load during pillar extraction. Mining geometry, MRS location, load histories, and roof fall locations are illustrated in figures 8, 9, and 10.

Pillar Failure Mechanism

Monitoring results during the extraction of the second half of pillar 1 clearly show the influence of fender failure and deterioration in roof stability. No roof falls occurred within the mining areas of interest while mining pillar 1 (figure 8). However, three roof falls occurred near the outside boundary of the excavation. These falls took place during and after extraction of cuts 11 and 12 when pillar failure was in progress as the effective pillar area was reduced. Maximum yield load was achieved on MRS 4 during excavation of the pushout cut, completing the pillar failure process. The instantaneous measured rates varied from 280 to 450 kPa/sec (40 to 65 psi/sec) during these events. Average load rates varied between 75 to 105 kN/min (17,000 to 24,000 lb/min) during this process.

Pillar Yielding Mechanism and Geologic Structure

Figure 9 presents mining geometry and roof fall locations during the extraction of pillar 2. Two roof falls occurred while completing approximately 50% extraction in the pillar. The first was outside of areas of active mining during mining cut 12. The second roof fall buried the continuous miner during mining of cut 14; this roof fall was structurally controlled by north-south-trending joints. At this time, the effective area of the pillar was reduced by approximately 50%, and thus the pillar approached yielding and the post-failure regime. This assertion is made based on stress analyses (figure 6) and loading

[1]Mention of specific products and manufacturers does not imply endorsement by the National Institute for Occupational Safety and Health.

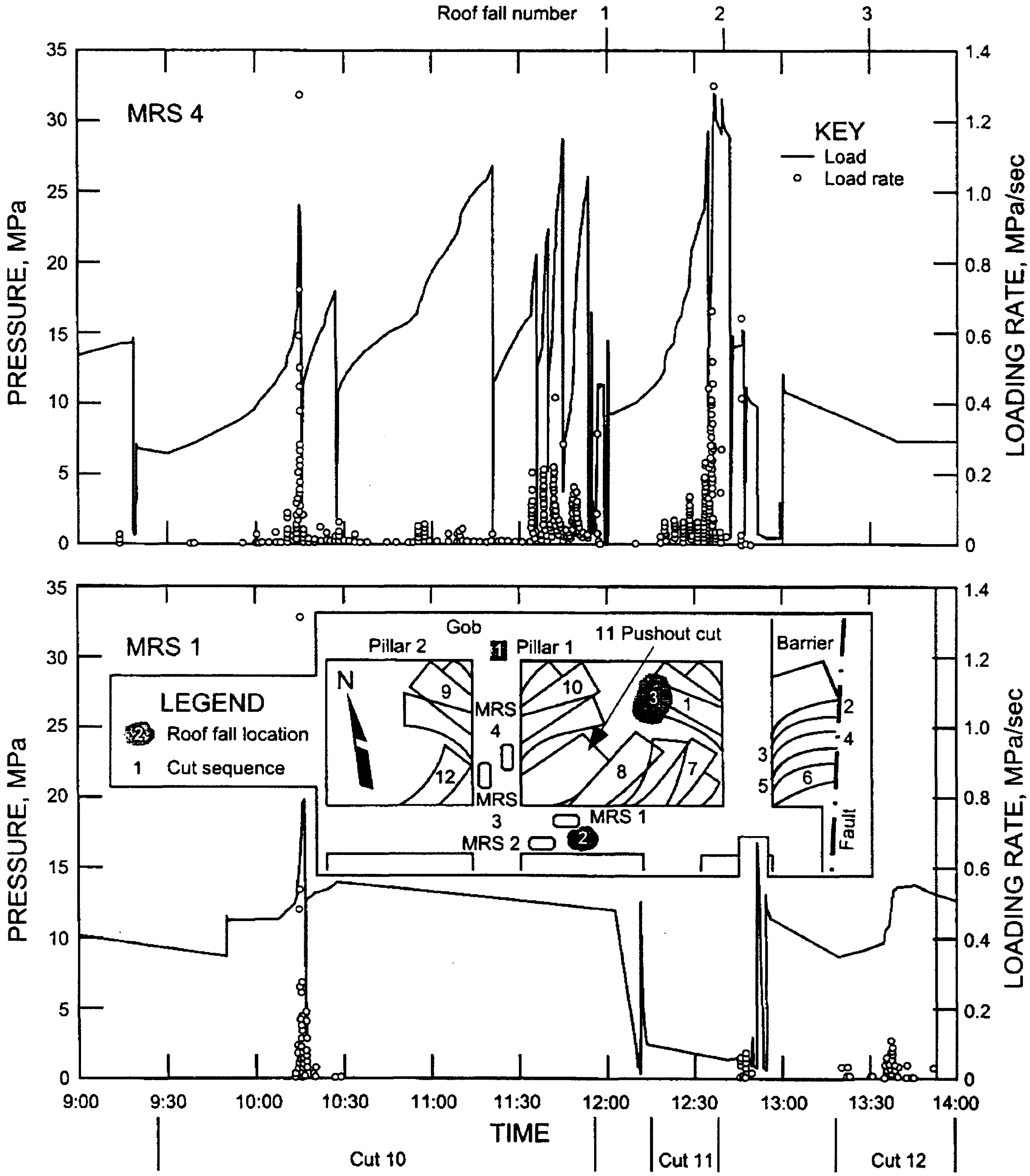

Figure 8.—Load history for MRS's 1 and 4 while completing extraction of pillar 1.

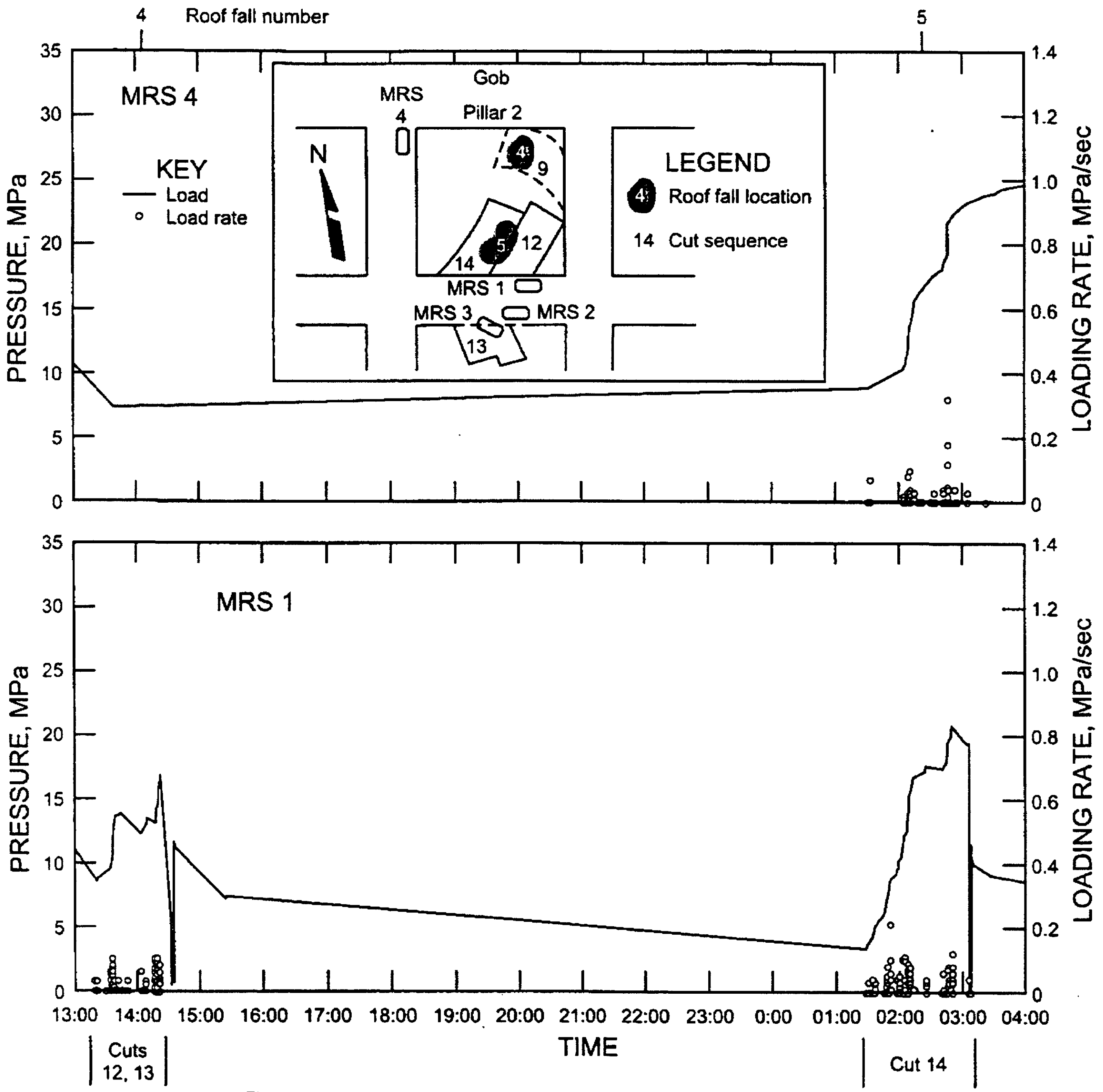

Figure 9.—Load history for MRS's 1 and 4 while completing extraction of pillar 2.

patterns on MRS's 1 and 4 (figure 9). Three minutes prior to this roof fall, load on both MRS 1 and 4 increased to approximately 21 MPa (3,000 psi) and instantaneous load rates varied from 120 to 200 kPa/sec (18 to 29 psi/sec) with an average load rate of 670 kN/min (15,000 lbf/min). In comparison to events for pillar 1, both load and load rates were lower because pillar 2 still provided sufficient resistance to limit roof-floor convergence. We suspect that pillar 2 was in a post-failure load deformation stage because there was a gradual load increase on MRS 4 during equipment recovery operations. At the termination of recovery operations, load was approaching 28 MPa (4,000 psi). Pillar yielding thus appear to have triggered movements in roof blocks outlined by preexisting joints.

Seismically Triggered Roof Falls

Figure 10 presents mining geometry and roof fall location during the extraction of pillar 8 and the barrier pillar. At this location, only MRS's 3 and 4 were used. One roof fall, a block of rock 1 by 1 by 3.7 m (3.3 by 3.3 by 12 ft), occurred during the extraction of cut 6. The block was structurally controlled by north-south-trending joints. At the time of failure, loads were moderate on both MRS 3 and 4 of about 21 MPa (3,000 psi) and instantaneous load rates were generally small, 76 kPa/sec (11 psi/sec) with average load rate of 36 kN/min (8,000 lbf/min). A large, instantaneous increase in the load rate of 390 kPa/sec (56 psi/sec) was measured on MRS 3 shortly before the block

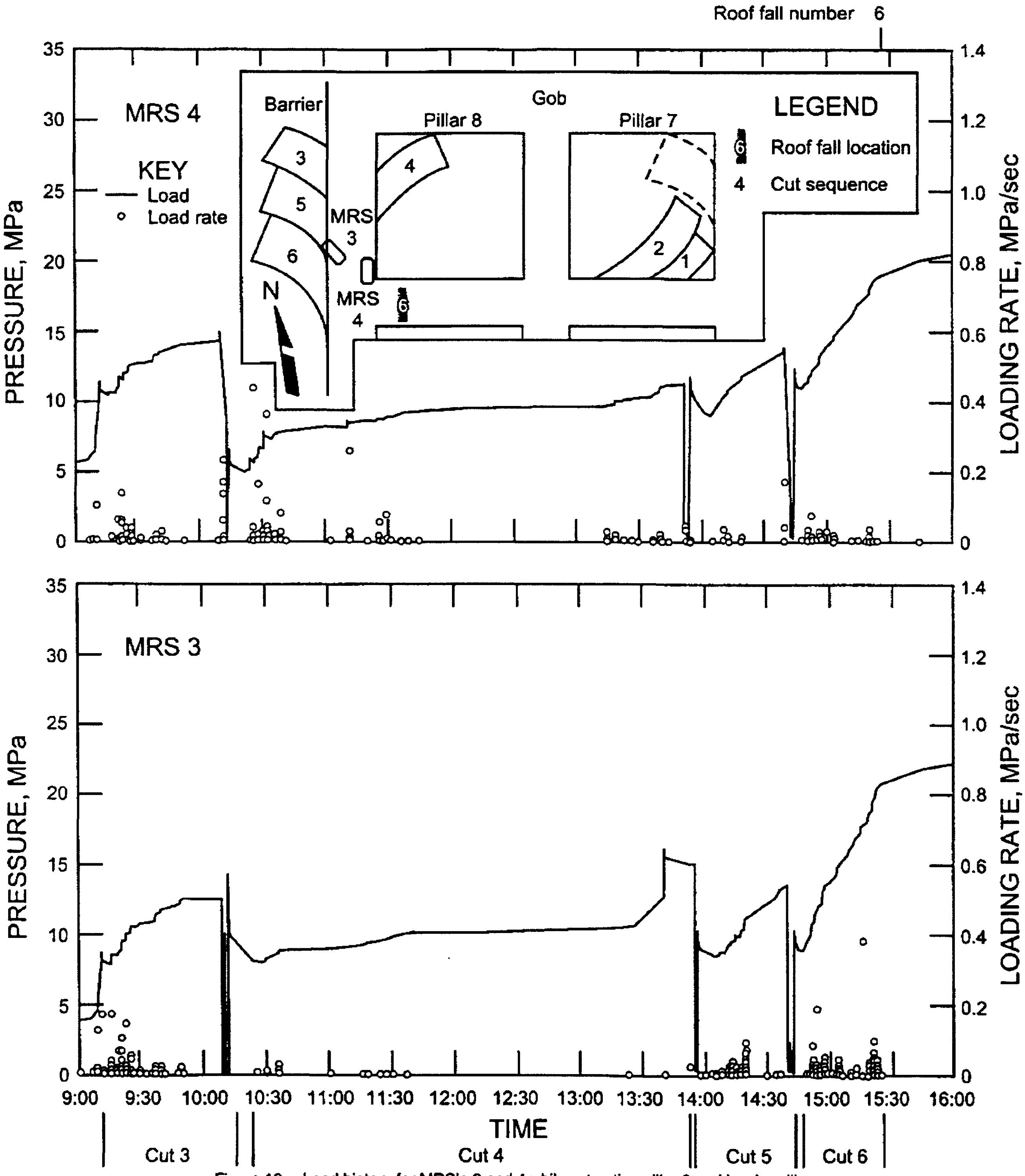

Figure 10.—Load history for MRS's 3 and 4 while extracting pillar 8 and barrier pillar.

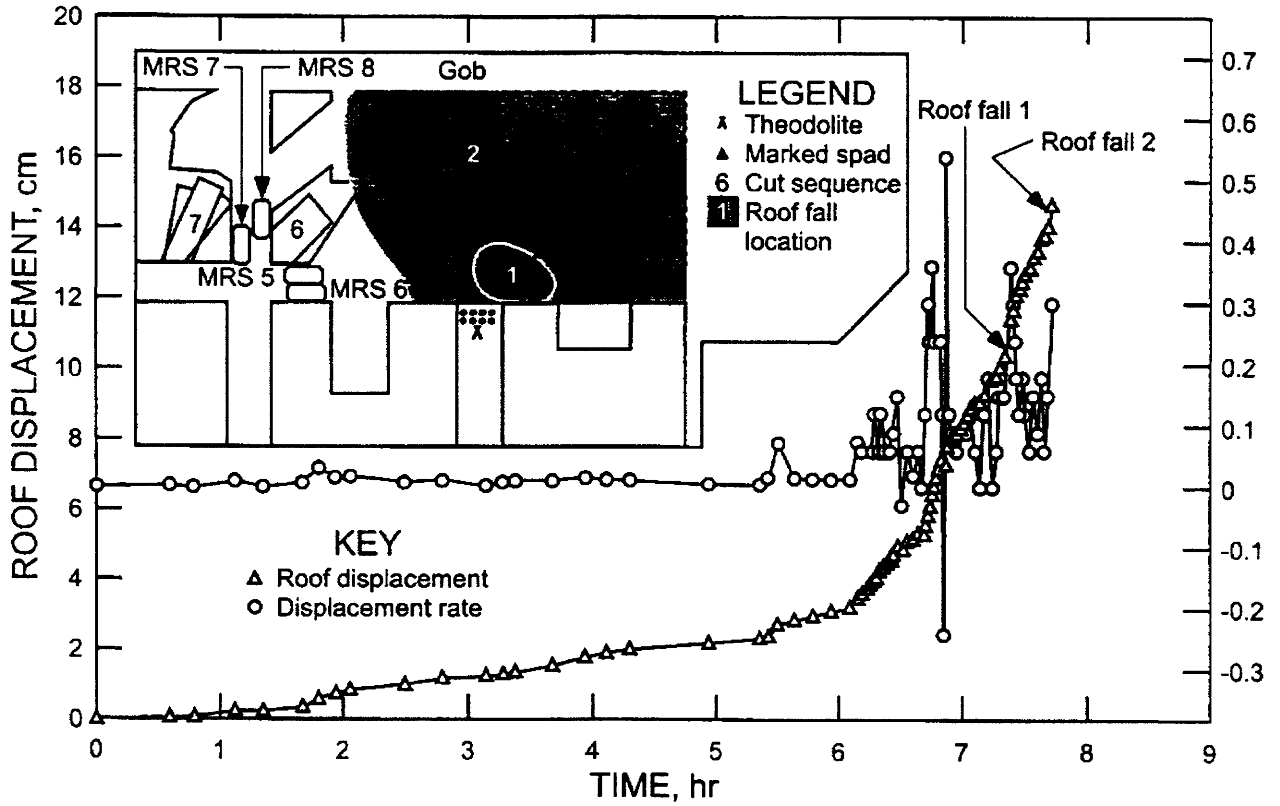

Figure 11.—Roof displacement history prior to major roof falls.

fell. The authors suspect preexisting structures to have contributed to this roof fall, which was triggered by higher-than-normal mine seismicity in this high-stress area.

Typical Deformation Monitoring Results

Reliable detection of impending stability problems requires monitoring both loading patterns on MRS's and roof movements. This is illustrated by presenting a roof deformation and major caving history approximately 30.5 m (100 ft) behind the face using a theodolite and marked spads at the study mine in the Blind Canyon Seam (figure 11). Note an increase in roof deformation and roof deformation rate prior to two roof falls. The deformation rate exceeded a critical rate of 0.5 cm/min (0.2 in/min) approximately 30 min prior to roof fall 1. This critical rate is in close agreement with other measurements of convergence in four other U.S. coal mines (Maleki 1981; Maleki 1988; Maleki et al. 1999). The second roof fall was associated with seismicity that was registered as spikes in the loading patterns on four MRS's.

CONCLUSIONS AND RECOMMENDED WORK

To eliminate setting and handling posts and reduce the number of miners required to work near the cave line and at other dangerous locations, a remotely controlled MRS has been developed and tested in the field. Optimum use of MRS's depends on careful panel designs, mine orientation, and primary support designs geared to expected geologic and stress conditions (Maleki and Owens 2001). MRS's have a limited zone of influence around them and thus can best be utilized in combination with other MRS's and in conjunction with ground monitoring systems.

An integrated ground monitoring system was tested in which the simplicity of deformation measurements were combined with more elaborate load rate monitoring on MRS leg cylinders. Analyses of field data show that roof instabilities are influenced by four mechanisms: (1) pillar failure, (2) pillar yielding, (3) mine seismicity, and (4) geologic structures. Pillar yielding and failure (unloading) and seismicity can be conveniently monitored by the load rate monitoring device, but to detect impending roof falls, additional deformation measurements directly within the cuts are needed.

Preliminary results show that roof falls occur when roof movements accelerate, reaching critical limits of 0.5 cm/min (0.2 in/min). Using average loading rates on MRS's at the study mine, there is a high potential for roof-pillar failure when the MRS loading rate increases beyond 44 kN/min (10,000 lbf/min). At such high loading rates, it is considered very likely for an MRS and/or the continuous miner to be buried during either mining or relocating the MRS. Between 22 to 44 kN/min (5,000-10,000 lbf/min), stability problems are still likely to pose some risk to equipment and worker safety. Structurally controlled instabilities play a bigger role at the lower end of this range, depending on mine seismicity, geology, and operating conditions. Below 22 kN/min (5,000 lbf/min), the likelihood of pillar stability problems is very low, and overall stability can be controlled through prudent support and excavation techniques.

Research continues in testing and evaluating MRS performance under dynamic loading conditions. The focus of future ground control research is to quantify and verify critical loading parameters that are indicative of impending stability problems under different geologic conditions.

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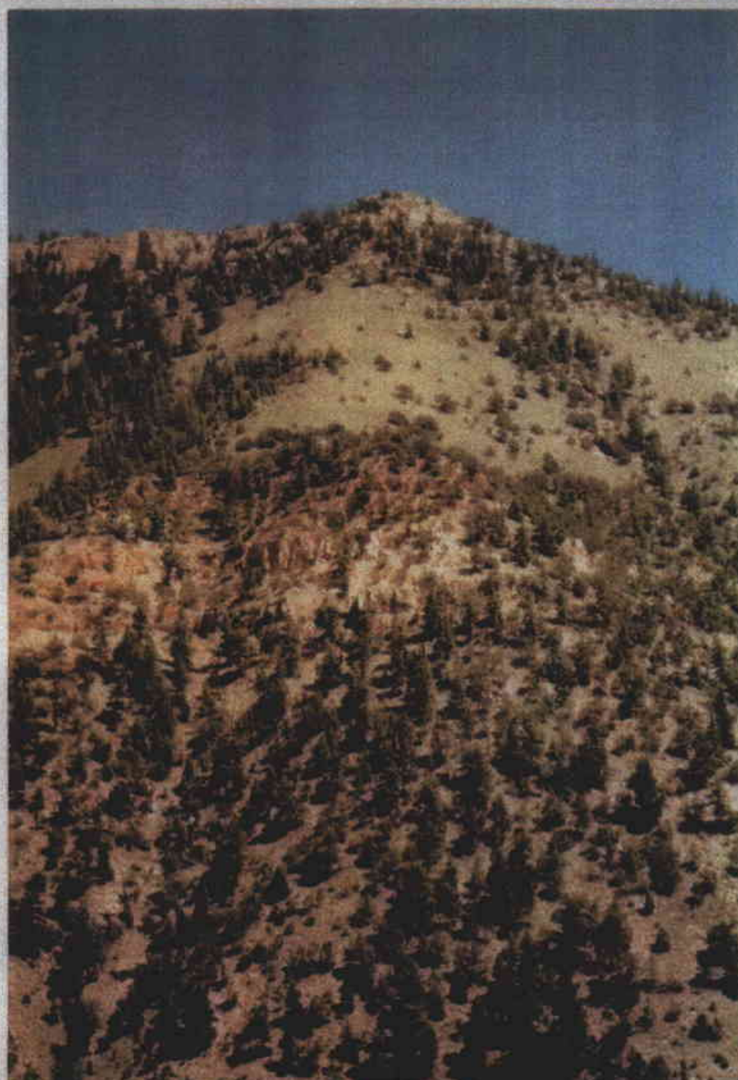
ATTACHMENT B

PHOTOGRAPHS OF CASTLEGATE ESCARPMENT CONDITIONS

Legend

Following 13 pages present typical conditions photographed from positions 1 to 13 (figure 2)



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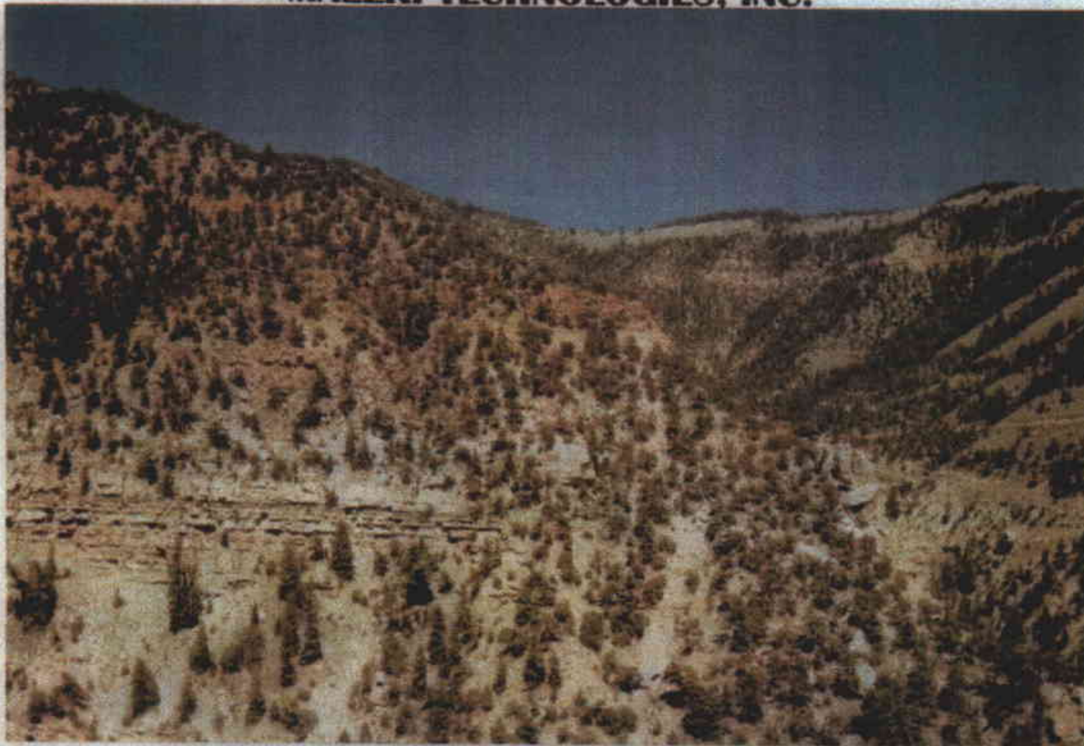

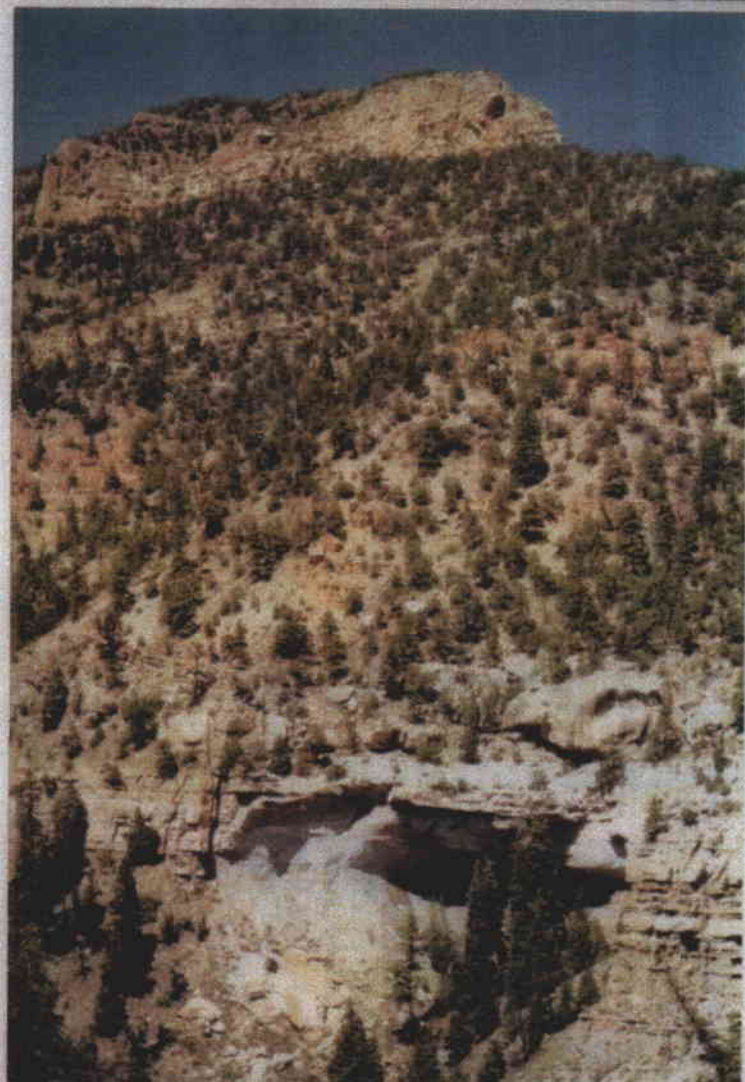

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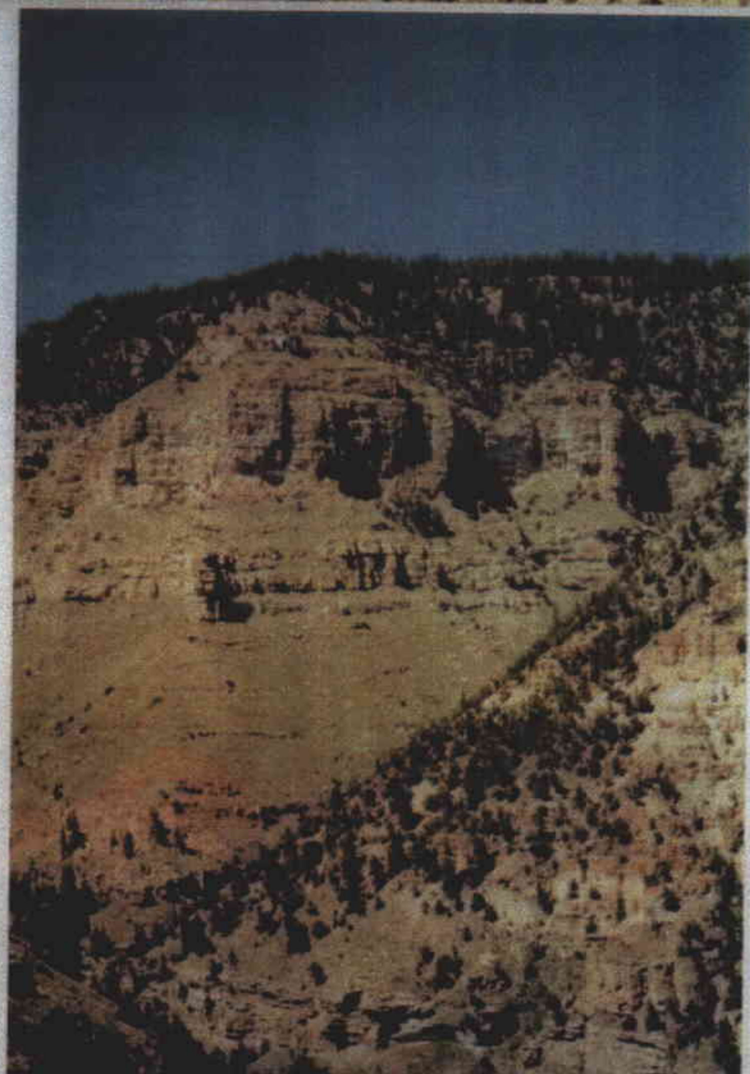

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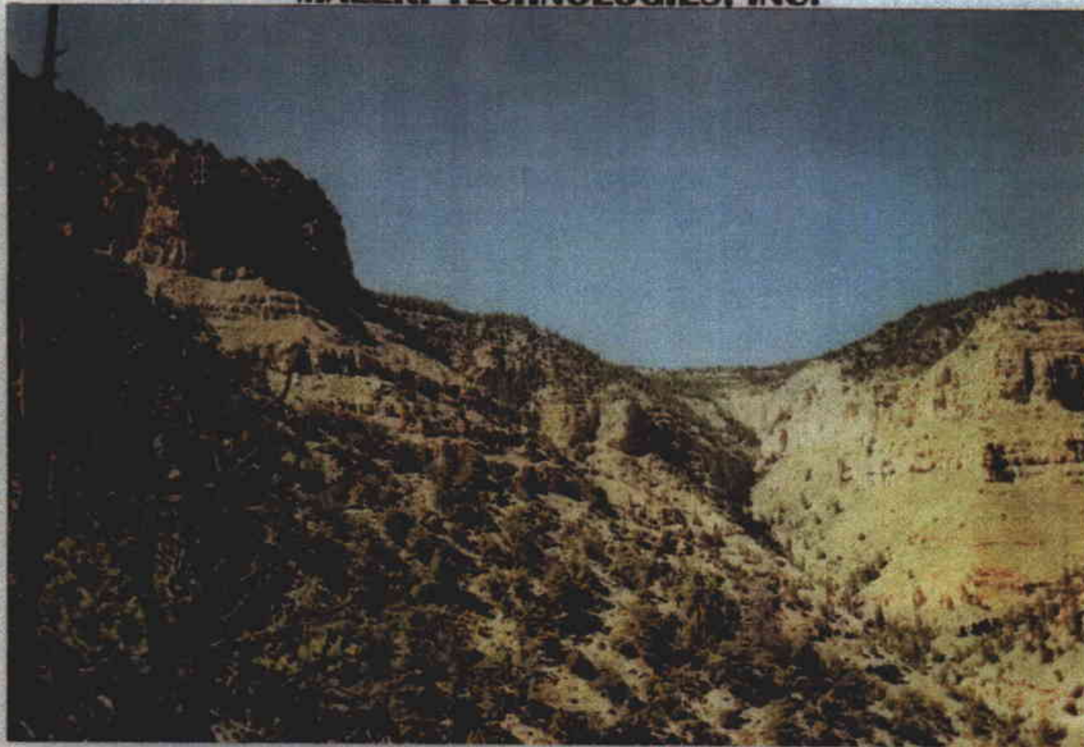

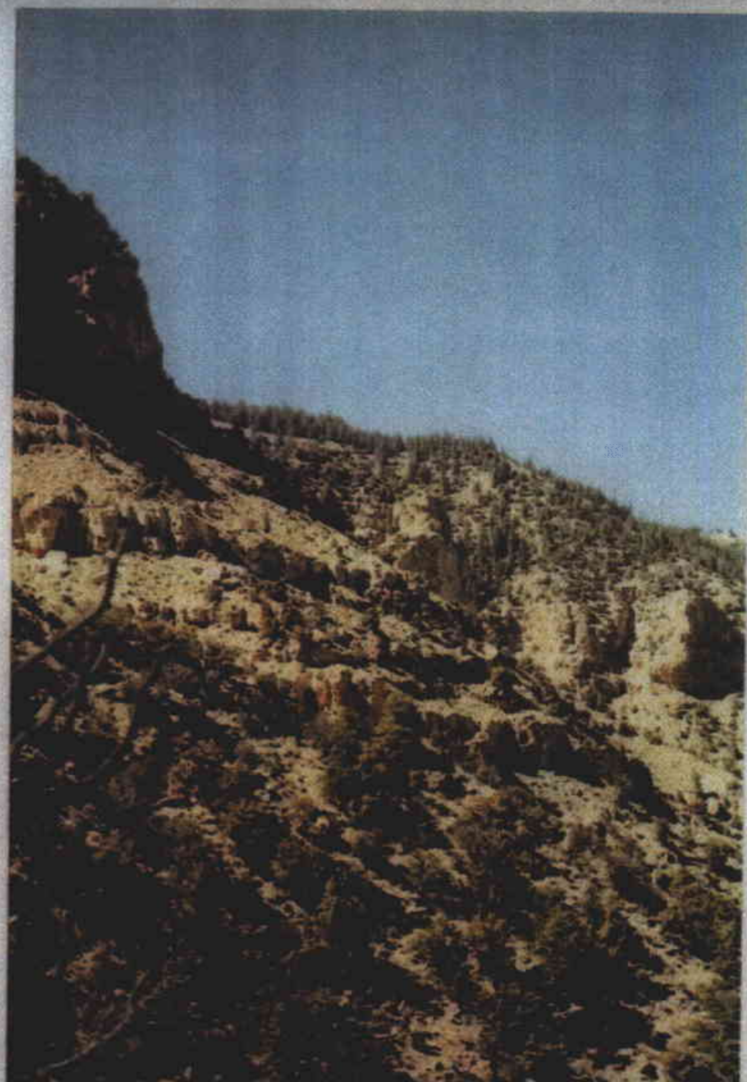

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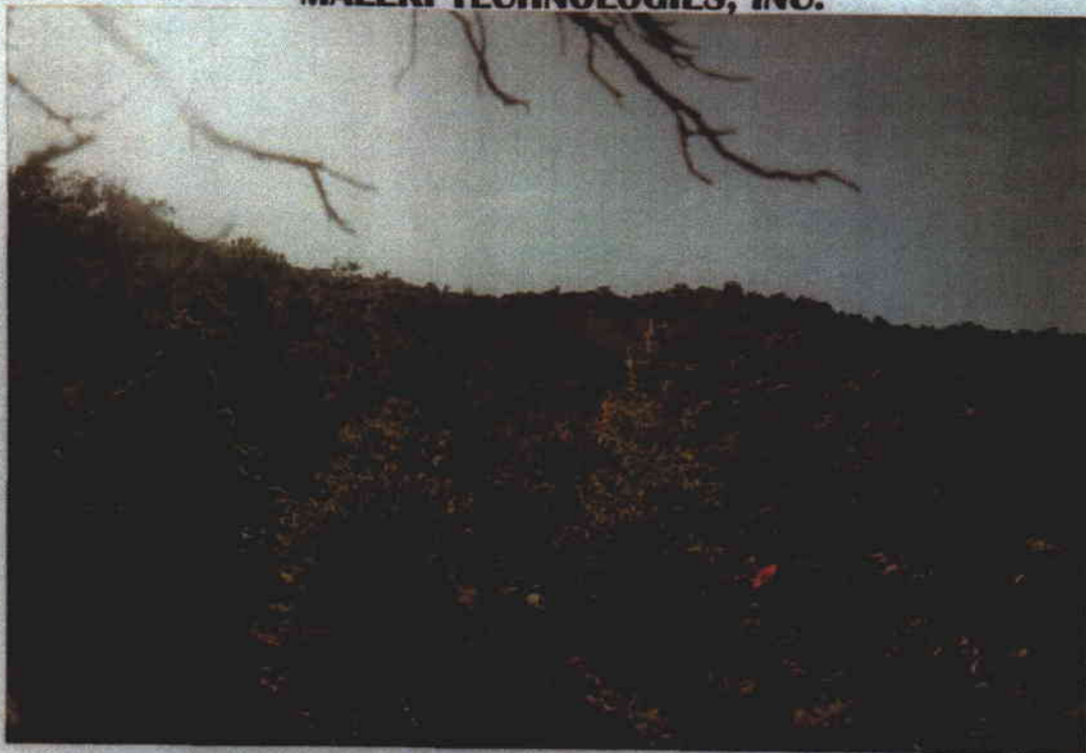

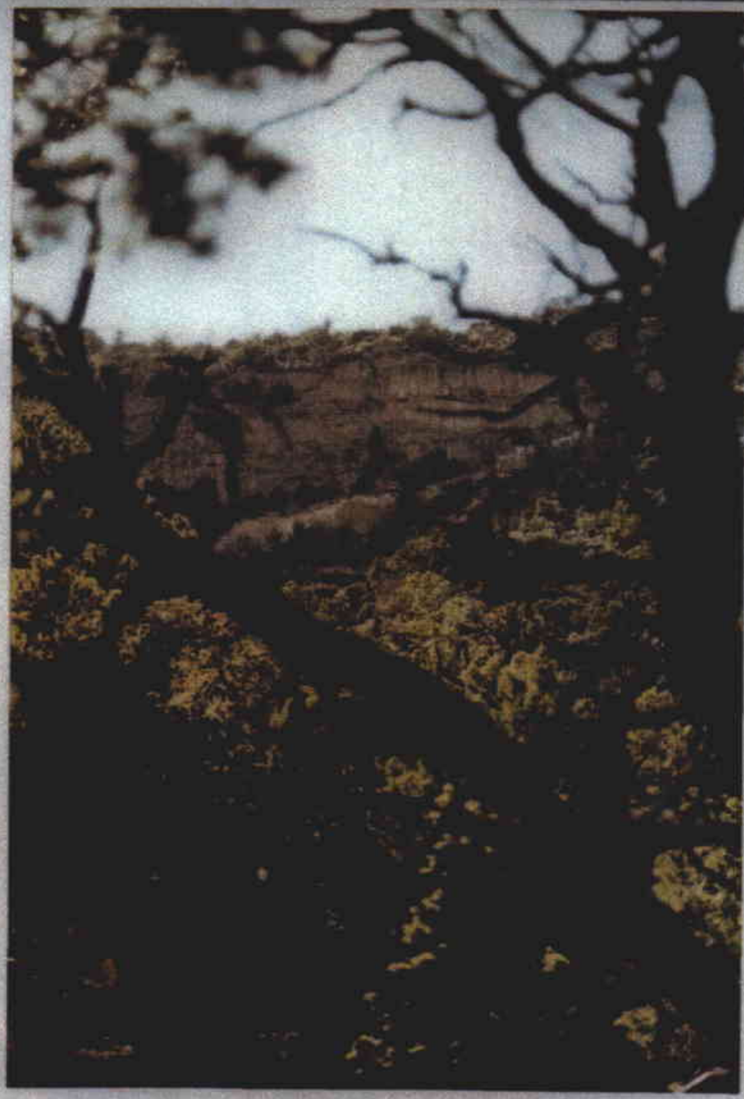

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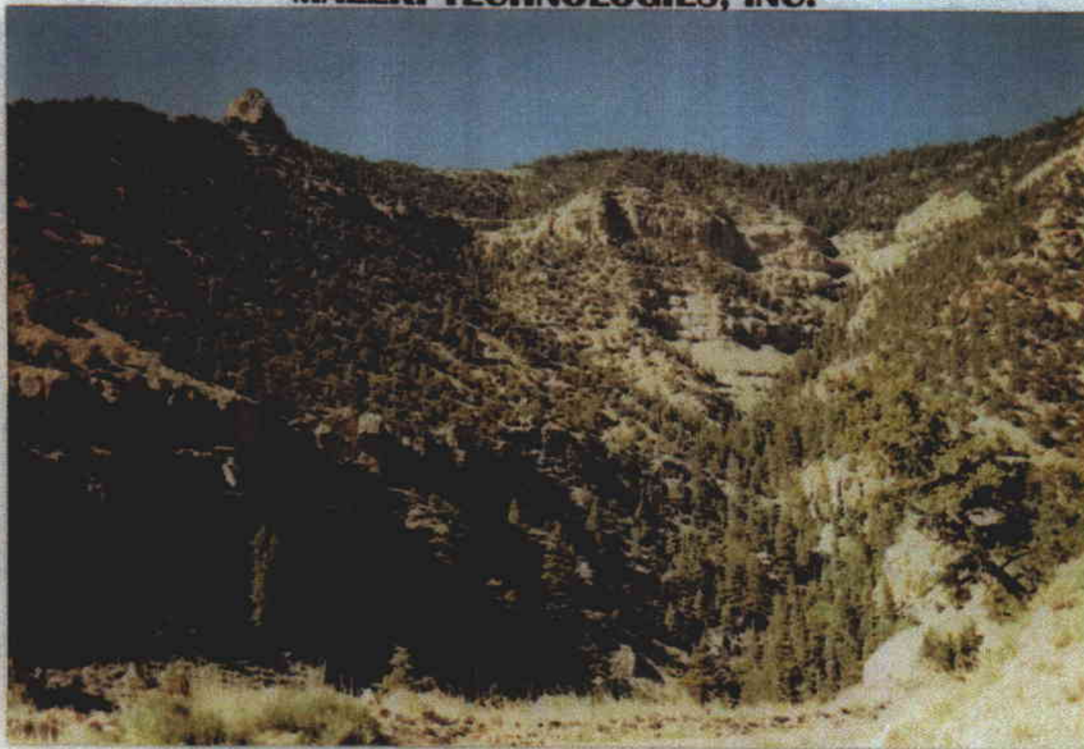
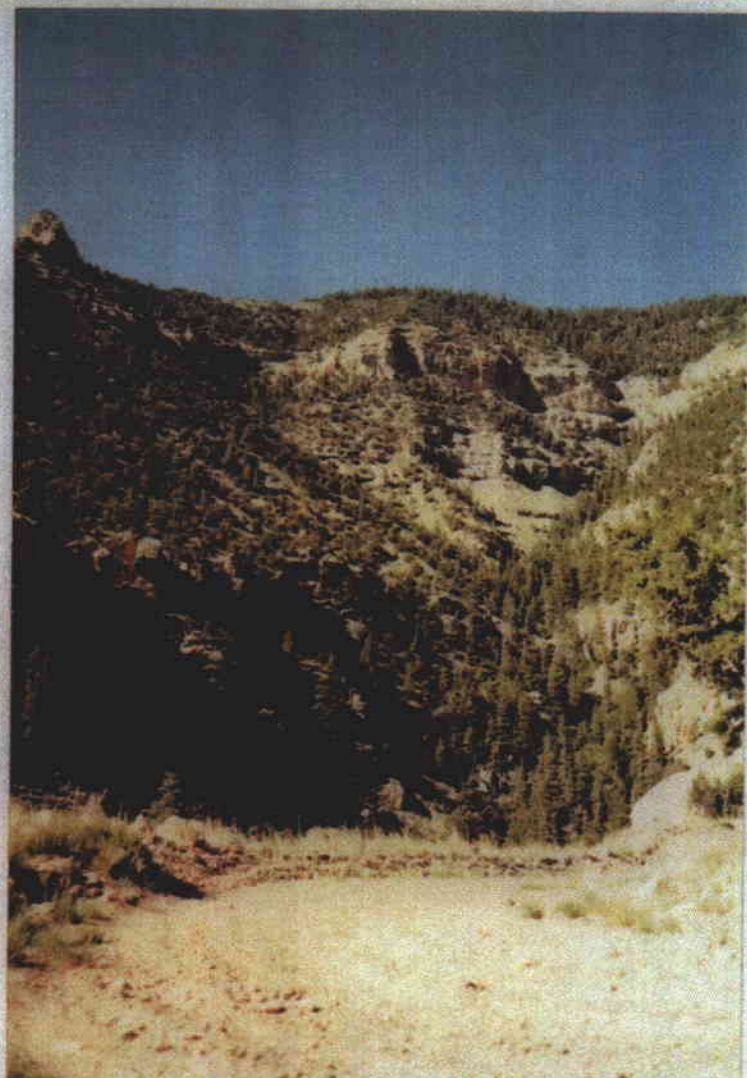
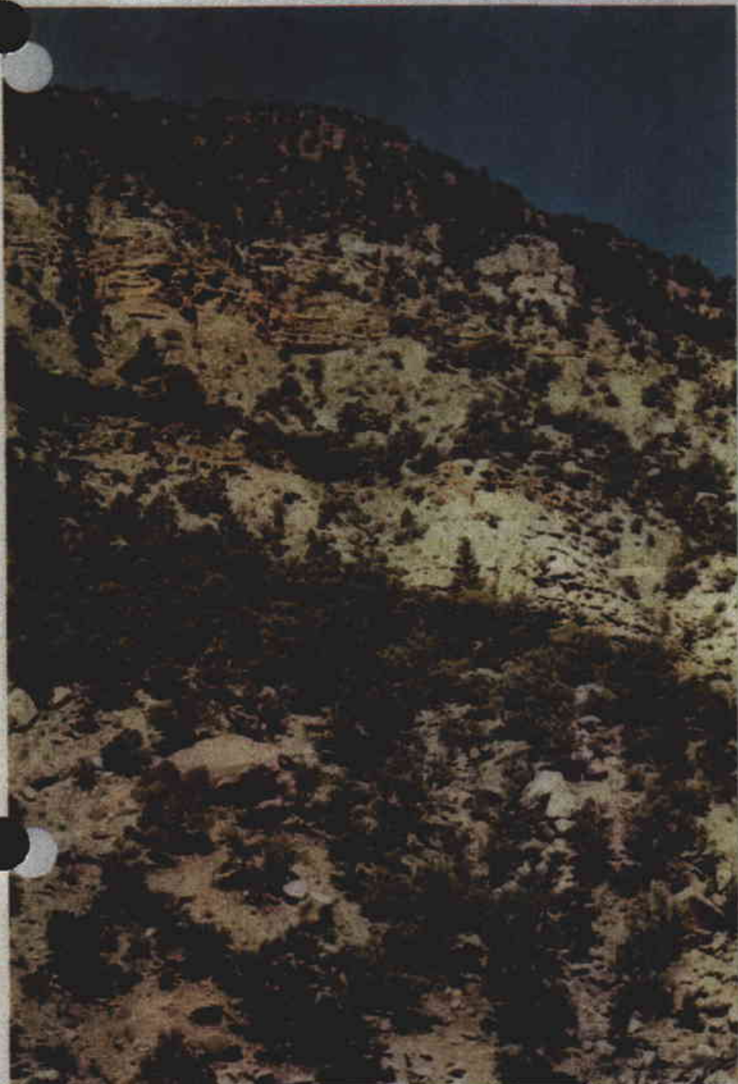
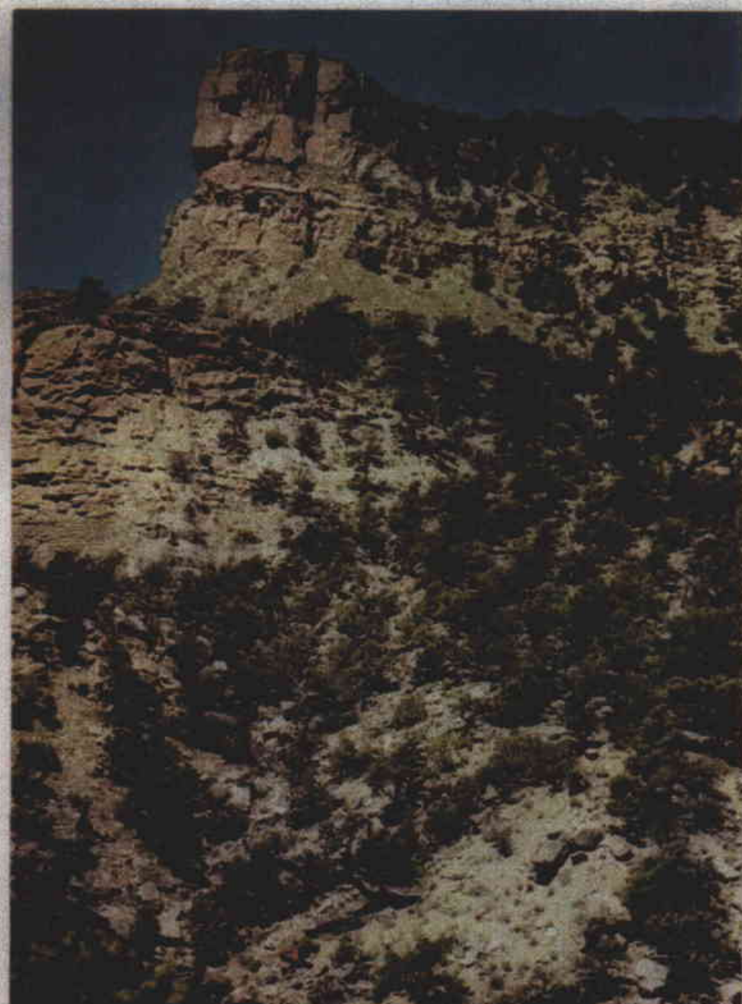
MALEKI TECHNOLOGIES, INC.









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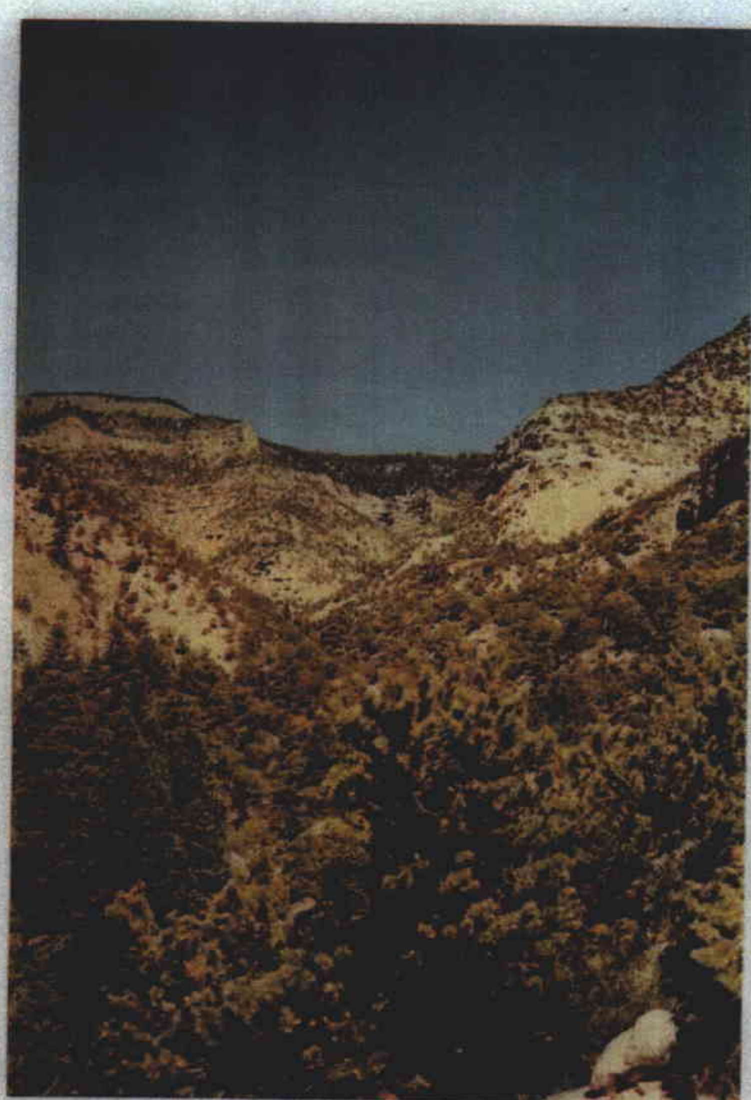

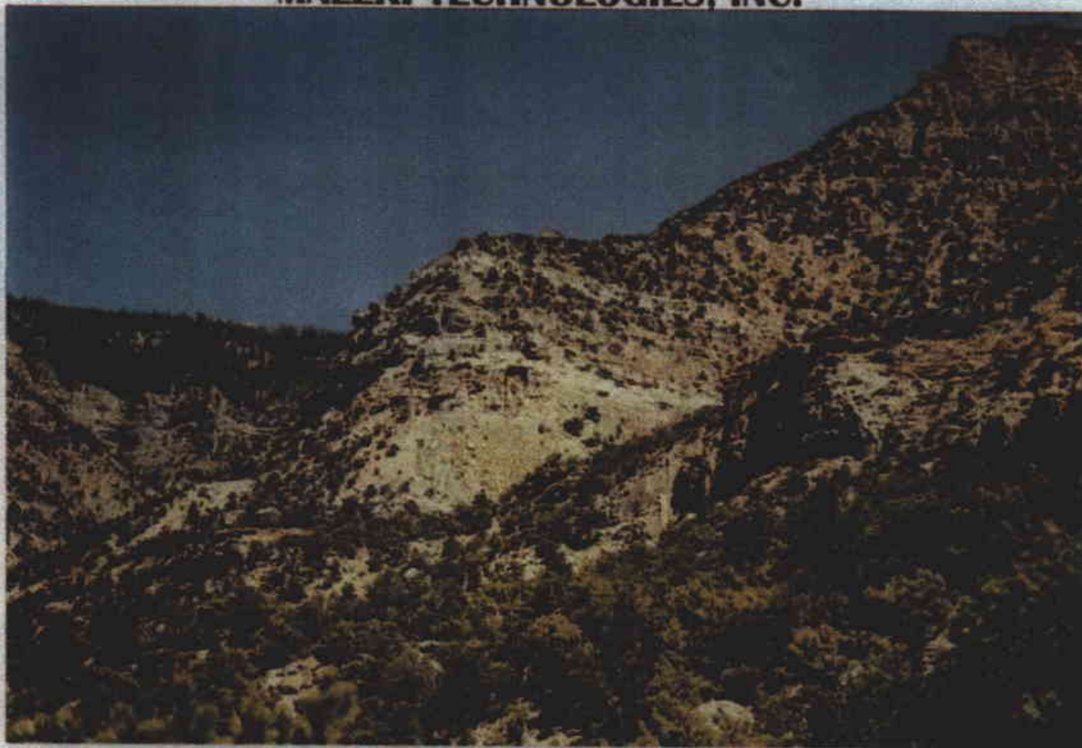

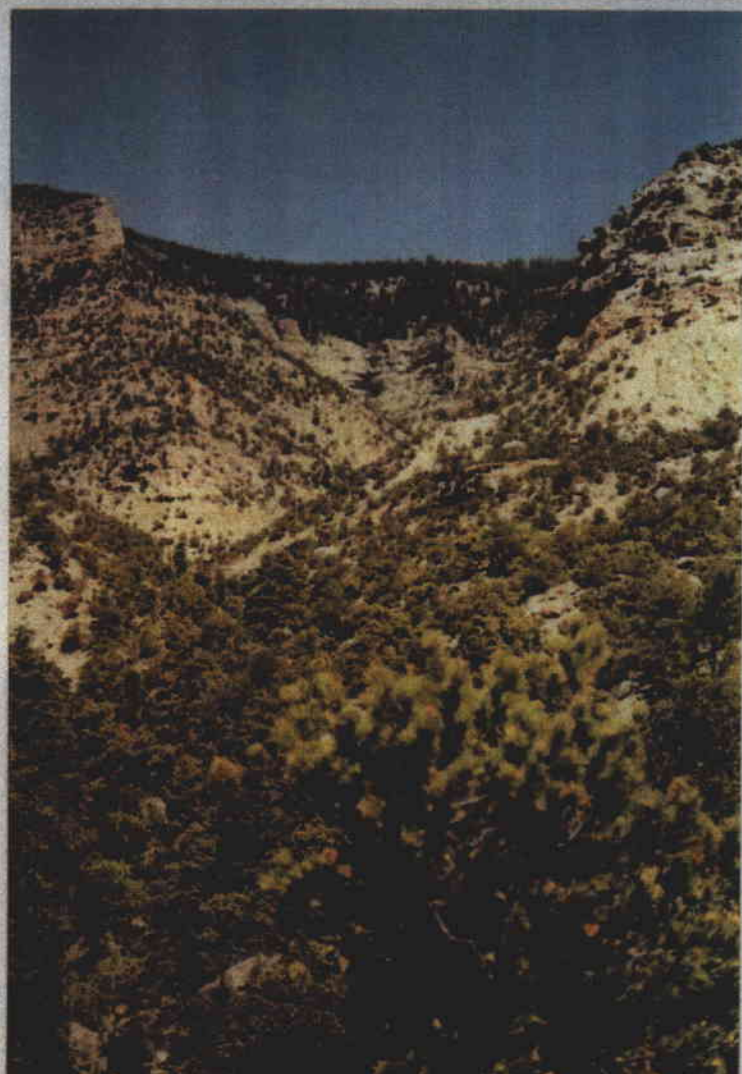

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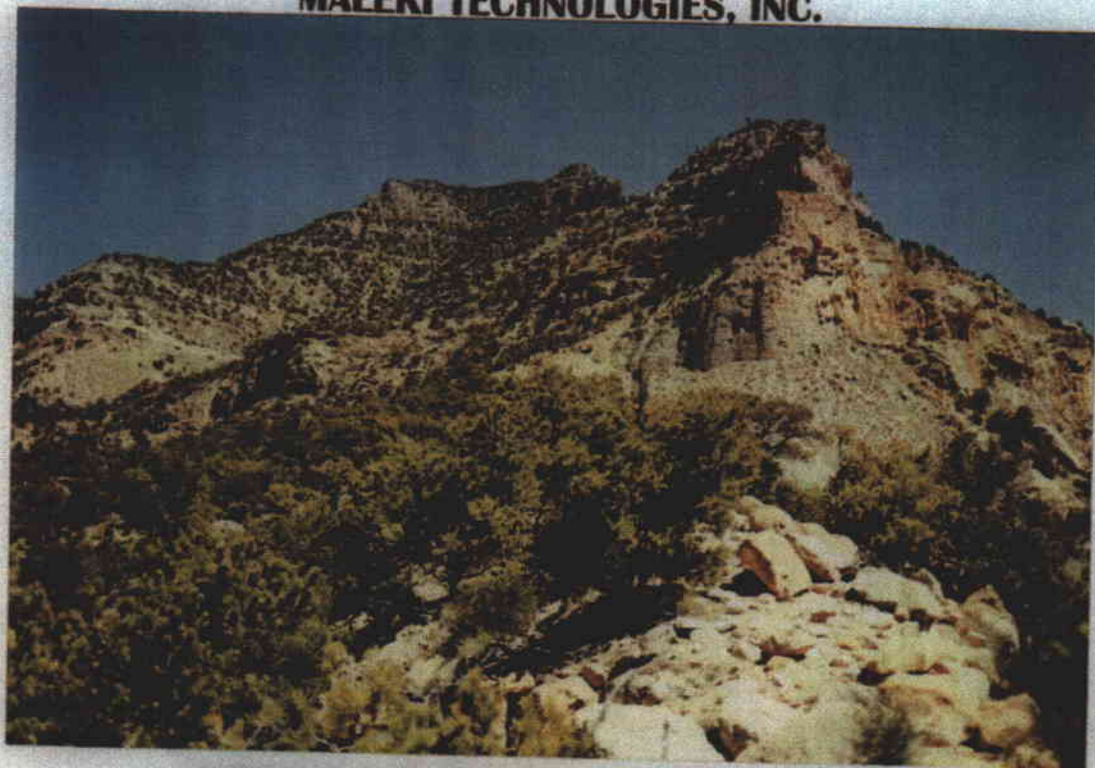
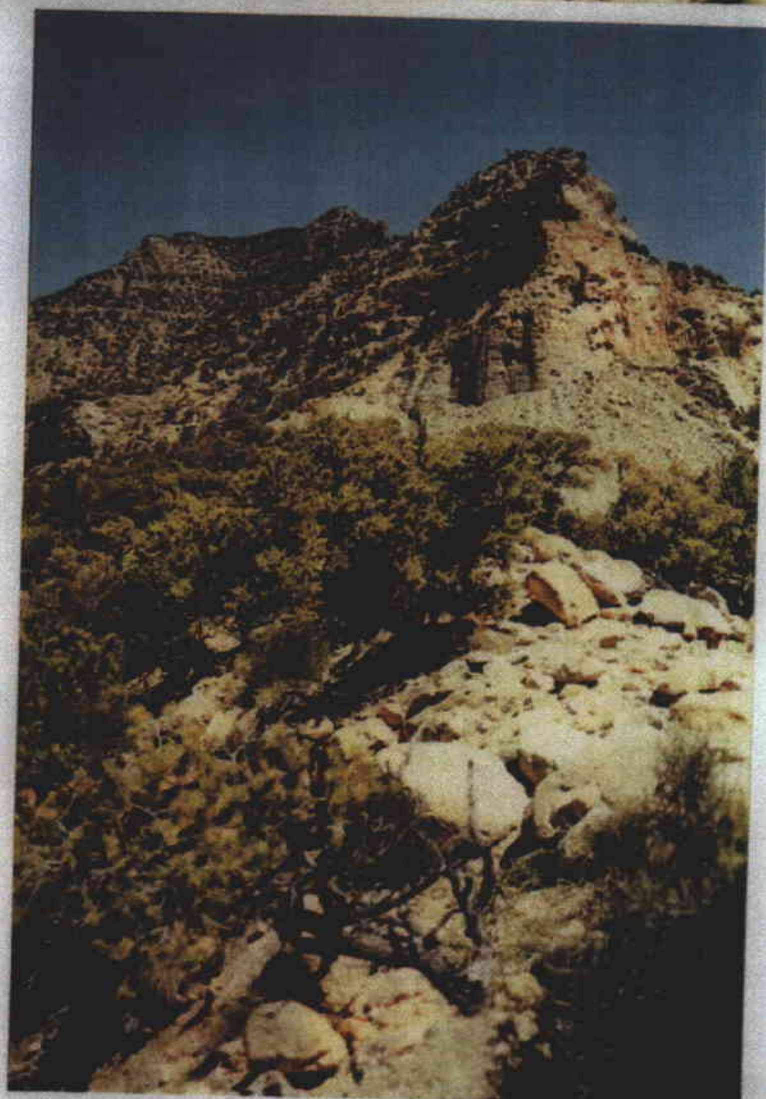
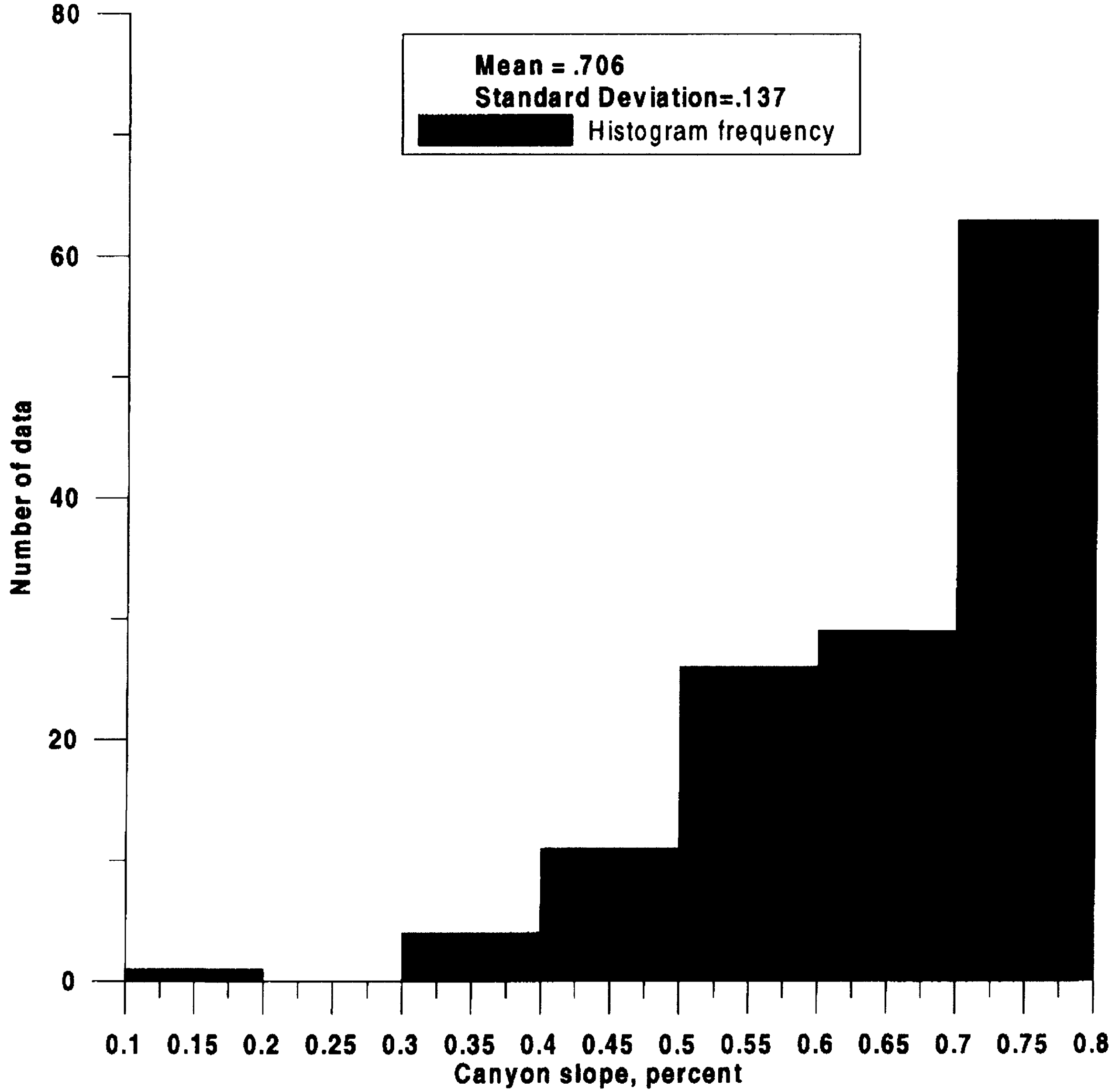
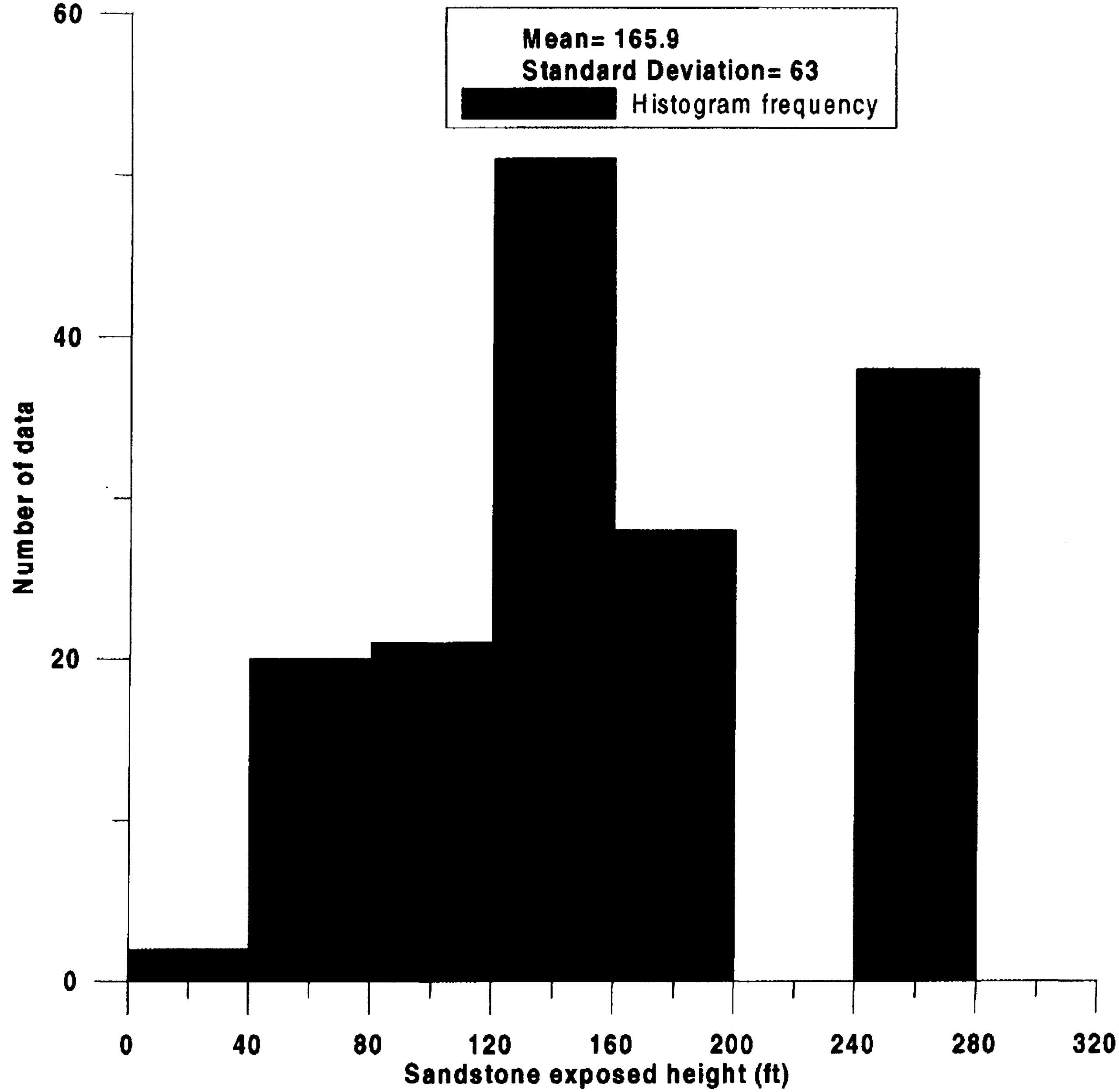




ATTACHMENT C

HISTOGRAM FREQUENCY DIAGRAMS FOR IMPORTANT VARIABLES


Mean = .706
Standard Deviation=.137
Histogram frequency
80
60
40
20
0
Number of data
0.1 0.15 0.2 0.25 0.3 0.35 0.4 0.45 0.5 0.55 0.6 0.65 0.7 0.75 0.8
Canyon slope, percent


Mean= 165.9
Standard Deviation= 63
Histogram frequency
Number of data
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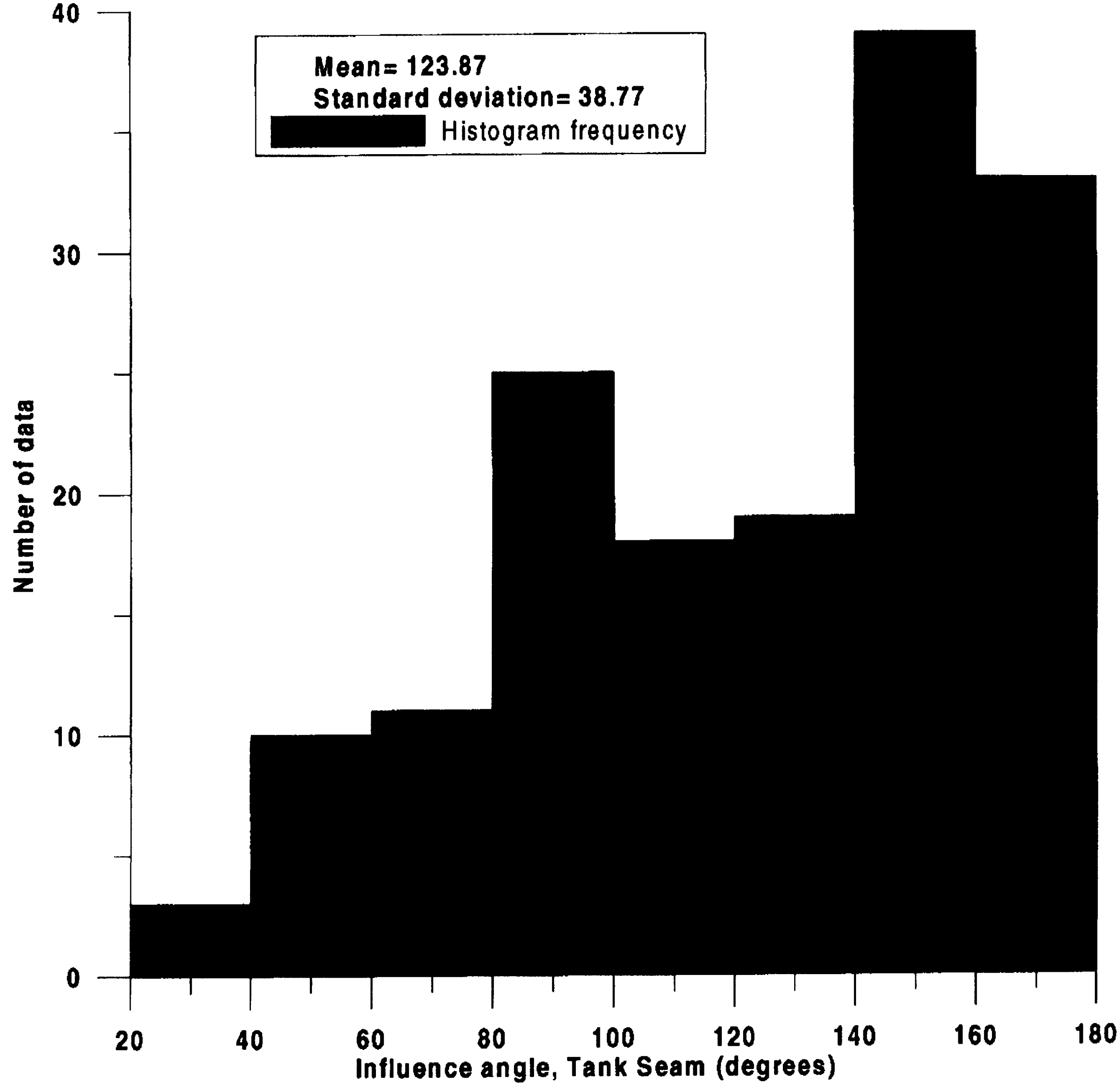
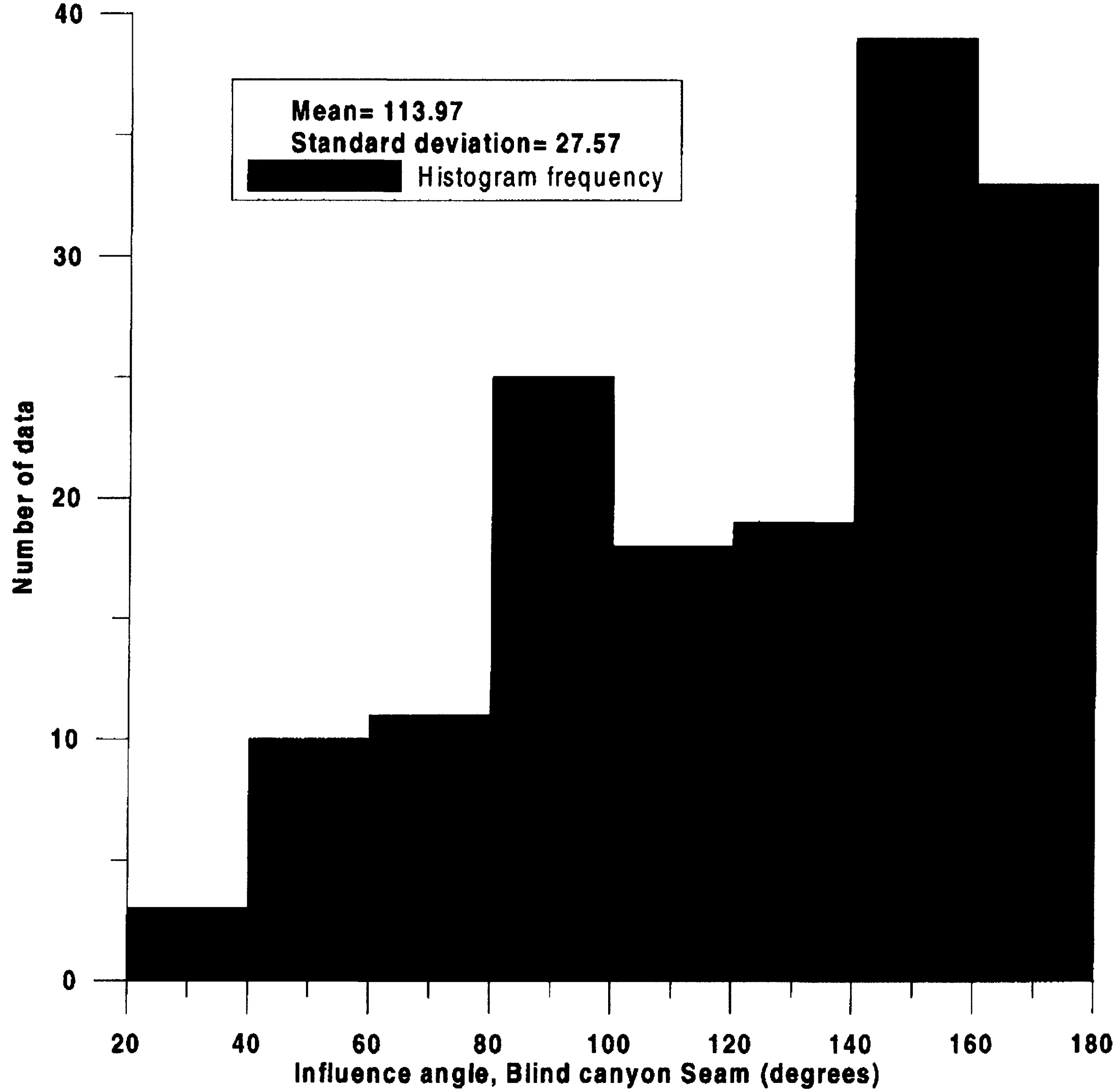
20
40
60
0
40
80
120
160
200
240
280
320
Sandstone exposed height (ft)



Mean= 123.87
Standard deviation= 38.77
Histogram frequency
Number of data
0
10
20
30
40
20
40
60
80
100
120
140
160
180
Influence angle, Tank Seam (degrees)



Mean= 113.97
Standard deviation= 27.57
Histogram frequency
Number of data
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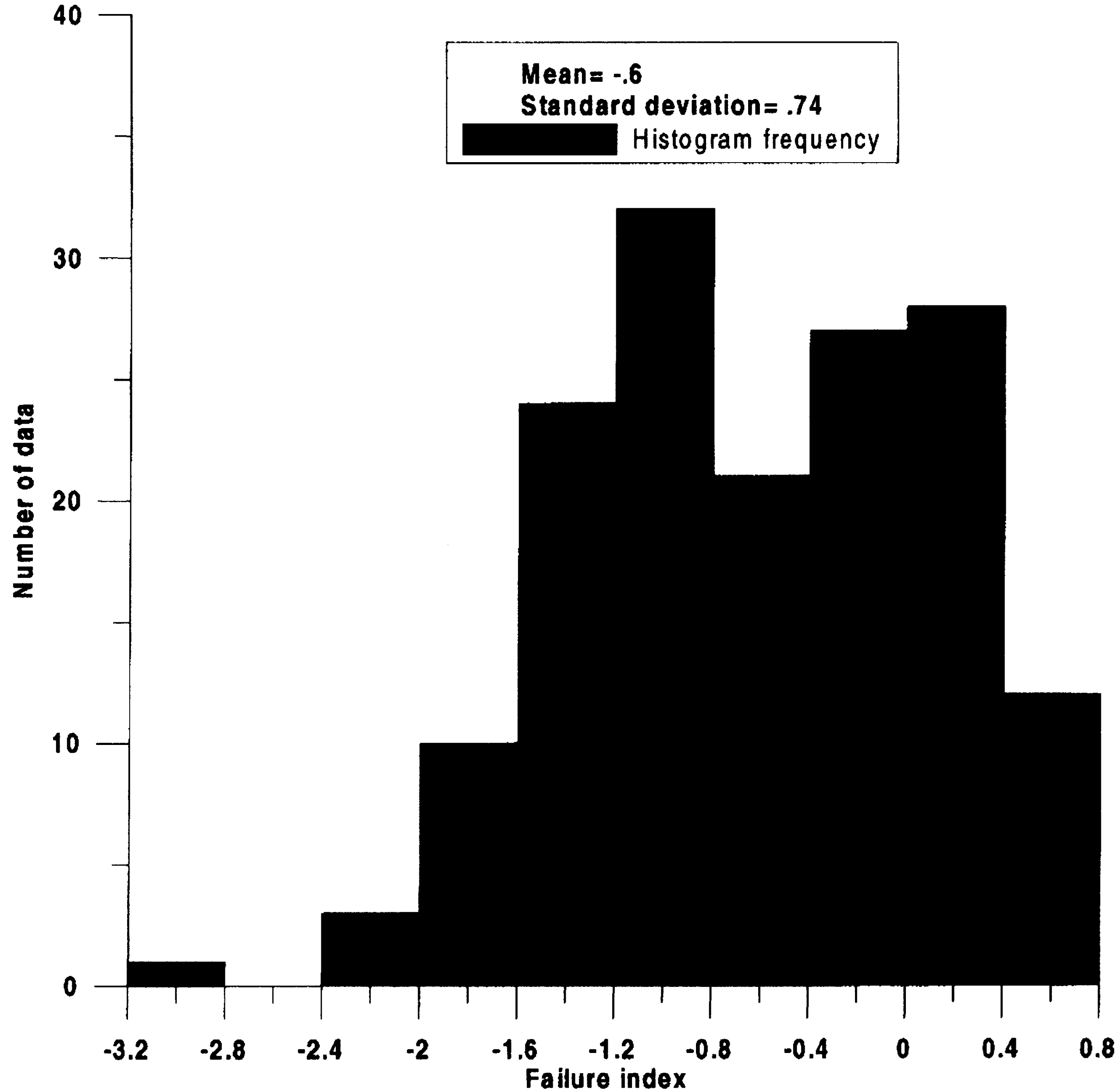
10
20
30
40
20
40
60
80
100
120
140
160
180
Influence angle, Blind canyon Seam (degrees)


Mean= -.6
Standard deviation= .74
Histogram frequency
Number of data
40
30
20
10
0
-3.2
-2.8
-2.4
-2
-1.6
-1.2
-0.8
-0.4
0
0.4
0.8
Failure index

ATTACHMENT D

PROGRAM OUTPUT FOR ROCK FALL SIMULATIONS

******************************* CRSP3 ******************************

COLORADO ROCKFALL SIMULATION PROGRAM

MODELS THE TRAJECTORY OF ROCKS ON IRREGULAR SLOPES
USING SLOPE PROFILE, SURFACE ROUGHNESS, SURFACE MATERIAL PROPERTIES
ROCK SIZE, AND ROCK SHAPE TO PRODUCE A STATISTICAL ANALYSIS OF
ROCKFALL BEHAVIOR ON THE SLOPE

****

BEARWEST.DAT

ROCK STATISTICS

18776 lbs SPHERICAL ROCK   3 ft RADIUS

NUMBER OF CELLS	9
NUMBER OF ROCKS	25
ANALYSIS POSITION	650 ft
INITIAL Y ZONE	8400 ft TO 8560 ft

INITIAL X VELOCITY	1 ft/sec
INITIAL Y VELOCITY	-1 ft/sec

CELL DATA TABLE

BEARWEST.DAT

REMARKS: BEAR CANYON WEST, BACK ANALYSIS

CELL#	SURFACE ROUGHNESS	TANGENTIAL COEFFICIENT	NORM. COEF. RESTITUTION	BEGINNING X,Y	ENDING X,Y
1	5.00	.85	.35	0.0 ,8800.0	350.0 ,8560.0
2	1.00	.87	.37	350.0 ,8560.0	650.0 ,8400.0
3	5.00	.85	.35	650.0 ,8400.0	700.0 ,8240.0
4	5.00	.87	.35	700.0 ,8240.0	800.0 ,8160.0
5	5.00	.82	.33	800.0 ,8160.0	1120.0 ,8000.0
6	5.00	.82	.33	1120.0 ,8000.0	1250.0 ,7920.0
7	5.00	.82	.33	1250.0 ,7920.0	1300.0 ,7840.0
8	5.00	.82	.33	1300.0 ,7840.0	1700.0 ,7600.0
9	5.00	.82	.33	1700.0 ,7600.0	2250.0 ,7440.0

ANALYSIS POINT DATA
BEARWEST.DAT

REMARKS: BEAR CANYON WEST, BACK ANALYSIS

ANALYSIS POINT . . . . X = 650 , Y =8400
TOTAL ROCKS PASSING ANALYSIS POINT 24

CUMULATIVE PROBABILITY	VELOCITY ft/sec	ENERGY ft - lbs	BOUNCE HEIGHT ft
50%	44.51	822308	0.58
75%	53.44	1108441	7.62
90%	61.47	1365800	13.95
95%	66.29	1520309	17.75
98%	71.70	1693718	22.01
	(NORMAL DISTRIBUTION)		(LOG DISTRIBUTION)

VELOCITY

BOUNCE HEIGHT

áááááááá

MAXIMUM	64.28 ft/sec
AVERAGE	44.51 ft/sec
MINIMUM	16.20 ft/sec
STANDARD DEVIATION	13.22 ft/sec

ááááááááááááá

MAXIMUM	5.97 ft
AVERAGE	1.68 ft
GEOMETRIC MEAN	0.58 ft
STANDARD DEVIATION	10.42 ft

KINETIC ENERGY
ááááááááááááááá

MAXIMUM	1482101 ft - lbs
AVERAGE	822308 ft - lbs
STANDARD DEVIATION	423776 ft - lbs

ANALYSIS POINT BOUNCE HEIGHT DISTRIBUTION

BEARWEST.DAT

```
BOUNCE
HEIGHT ft

  6Ç Ô
  5Ç
  4Ç ÔÔÔ
  3Ç ÔÔÔ
  2Ç ÔÔÔÔ
  1Ç ÔÔÔÔÔÔ
  0Ç ÔÔÔÔÔÔÔ
     0          10         20         30         40         50         60

        FREQUENCY
```

ANALYSIS POINT VELOCITY DISTRIBUTION

BEARWEST.DAT

```
FREQUENCY

 2Ç                        Ü         Ü         Ü    Ü
 1Ç Ü     Ü   Ü  Ü   Ü Ü  ÜÜÜ Ü    ÜÜÜÜ Ü  Ü Ü Ü  Ü Ü
     16                      40                       64

         VELOCITY ft/sec
```

BOUNCE HEIGHT GRAPH

BEARWEST.DAT

```
BOUNCE
HEIGHT ft

145Ç
137Ç                Ü
129Ç                ÜÜ
121Ç                ÜÜ
113Ç                ÜÜ
105Ç                ÜÜÜ
 97Ç                ÜÜÜ
 89Ç                ÜÜÜ
 81Ç                ÜÜÜ
 73Ç                ÜÜÜ
 65Ç               ÜÜÜÜÜ
 57Ç               ÜÜÜÜÜ
 49Ç               ÜÜÜÜÜ
 41Ç               ÜÜÜÜÜ
 33Ç               ÜÜÜÜÜ
 25Ç               ÜÜÜÜÜ
 17Ç               ÜÜÜÜÜ
  9Ç               ÜÜÜÜÜÜÜÜÜ Ü Ü
  1Ç       ÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜ
     0        374       749       1124       1499       1874       2249

          HORIZONTAL DISTANCE ft
```

VELOCITY GRAPH
BEARWEST.DAT

VELOCITY ft/sec

```
 147Ç
 139Ç                   ÜÜ
 131Ç                  ÜÜÜ
 123Ç                  ÜÜÜ
 115Ç                 ÜÜÜÜ
 107Ç                ÜÜÜÜÜ
  99Ç                ÜÜÜÜÜ
  91Ç                ÜÜÜÜÜ
  83Ç                ÜÜÜÜÜ
  75Ç                ÜÜÜÜÜ
  67Ç               ÜÜÜÜÜÜÜÜÜ
  59Ç             ÜÜÜÜÜÜÜÜÜÜÜ

      0         374       749       1124      1499      1874      2249

                HORIZONTAL DISTANCE ft
```

CELL DATA OUTPUT

BEARWEST.DAT

REMARKS: BEAR CANYON WEST, BACK ANALYSIS

DATA COLLECTED AT END OF EACH CELL

CELL #	MAXIMUM VELOCITY (ft/sec)	AVERAGE VELOCITY	STANDARD DEVIATION VELOCITY	AVERAGE BOUNCE HEIGHT (ft)	MAXIMUM BOUNCE HEIGHT (ft)
1	NO ROCKS PASSED POINT				
2	64	44	13.21	1	6
3	97	75	10.50	97	133
4	134	65	41.06	14	53
5	45	35	0.00	5	8
6	NO ROCKS PASSED POINT				
7	NO ROCKS PASSED POINT				
8	NO ROCKS PASSED POINT				
9	NO ROCKS PASSED POINT				

X INTERVAL	ROCKS STOPPED
0 ft TO 10 ft	1
740 ft TO 750 ft	1
810 ft TO 820 ft	1
820 ft TO 830 ft	1
840 ft TO 850 ft	2
850 ft TO 860 ft	1
860 ft TO 870 ft	1
880 ft TO 890 ft	2

910 ft	TO	920 ft	1
920 ft	TO	930 ft	1
940 ft	TO	950 ft	3
960 ft	TO	970 ft	1
970 ft	TO	980 ft	1
980 ft	TO	990 ft	1
1000 ft	TO	1010 ft	1
1030 ft	TO	1040 ft	1
1040 ft	TO	1050 ft	1
1050 ft	TO	1060 ft	1
1080 ft	TO	1090 ft	1
1190 ft	TO	1200 ft	2

******************************** CRSP3 ******************************

COLORADO ROCKFALL SIMULATION PROGRAM

MODELS THE TRAJECTORY OF ROCKS ON IRREGULAR SLOPES
USING SLOPE PROFILE, SURFACE ROUGHNESS, SURFACE MATERIAL PROPERTIES
ROCK SIZE, AND ROCK SHAPE TO PRODUCE A STATISTICAL ANALYSIS OF
ROCKFALL BEHAVIOR ON THE SLOPE

****

WHR.DAT

ROCK STATISTICS

18776 lbs SPHERICAL ROCK   3 ft RADIUS

NUMBER OF CELLS	13
NUMBER OF ROCKS	25
ANALYSIS POSITION	800 ft
INITIAL Y ZONE	8160 ft TO 8400 ft

INITIAL X VELOCITY	1 ft/sec
INITIAL Y VELOCITY	-1 ft/sec

CELL DATA TABLE

WHR.DAT

REMARKS: SOUTH WILD HORSE RIDGE, B-B'

CELL#	SURFACE ROUGHNESS	TANGENTIAL COEFFICIENT	NORM. COEF. RESTITUTION	BEGINNING X,Y	ENDING X,Y
1	5.00	.85	.35	0.0 ,8800.0	100.0 ,8780.0
2	5.00	.85	.35	100.0 ,8800.0	300.0 ,8720.0
3	5.00	.85	.35	300.0 ,8720.0	400.0 ,8640.0
4	5.00	.85	.35	400.0 ,8640.0	500.0 ,8560.0
5	5.00	.85	.35	500.0 ,8560.0	600.0 ,8400.0
6	1.00	.87	.37	600.0 ,8400.0	800.0 ,8160.0
7	5.00	.85	.35	800.0 ,8160.0	1050.0 ,8000.0
8	5.00	.87	.35	1050.0 ,8000.0	1350.0 ,7840.0
9	5.00	.82	.33	1350.0 ,7840.0	1600.0 ,7680.0
10	5.00	.82	.33	1600.0 ,7680.0	1800.0 ,7520.0
11	5.00	.82	.33	1800.0 ,7520.0	1820.0 ,7440.0
12	5.00	.82	.33	1820.0 ,7440.0	2000.0 ,7360.0
13	5.00	.82	.33	2000.0 ,7360.0	2300.0 ,7280.0

ANALYSIS POINT DATA
WHR.DAT

REMARKS: SOUTH WILD HORSE RIDGE, B-B'

ANALYSIS POINT . . . . X = 800 , Y =8160
TOTAL ROCKS PASSING ANALYSIS POINT 24

CUMULATIVE PROBABILITY	VELOCITY ft/sec	ENERGY ft - lbs	BOUNCE HEIGHT ft
50%	64.28	1638737	2.38
75%	77.40	2208965	6.67
90%	89.19	2721851	10.52
95%	96.27	3029768	12.84
98%	104.22	3375352	15.44
	(NORMAL DISTRIBUTION)		(LOG DISTRIBUTION)

VELOCITY
ááááááááá

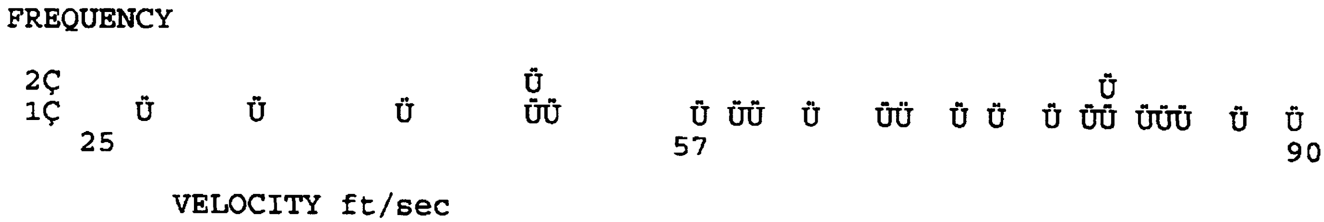
MAXIMUM 89.92 ft/sec
AVERAGE 64.28 ft/sec
MINIMUM 20.62 ft/sec

STANDARD
DEVIATION 19.43 ft/sec

BOUNCE HEIGHT
áááááááááááááá

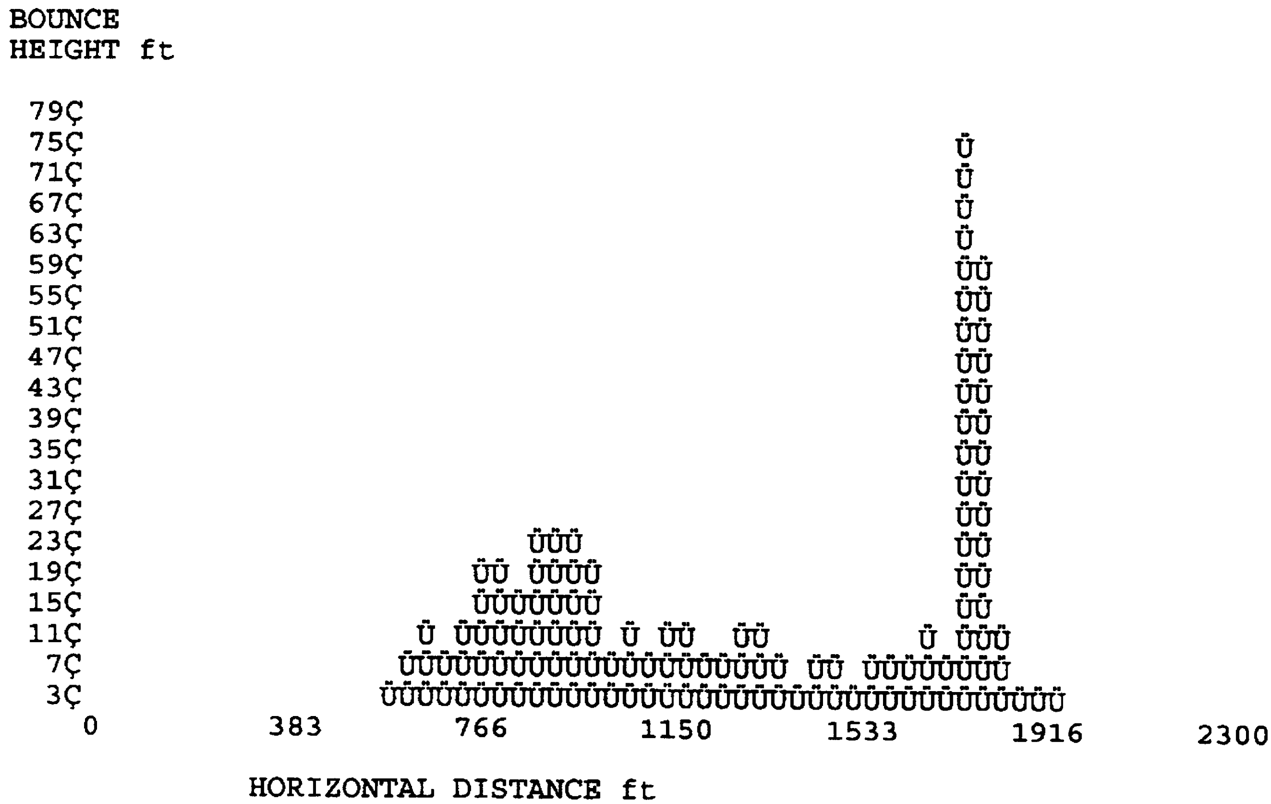
MAXIMUM 13.62 ft
AVERAGE 4.45 ft
GEOMETRIC MEAN 2.38 ft

STANDARD
DEVIATION 6.35 ft



BOUNCE HEIGHT GRAPH

WHR.DAT



VELOCITY GRAPH
WHR.DAT

VELOCITY ft/sec

```
96Ç
90Ç                    ÜÜ  Ü  Ü                    ÜÜ
84Ç                   ÜÜÜ ÜÜ ÜÜ                    ÜÜ
78Ç                  ÜÜÜÜÜÜÜÜÜÜ                   ÜÜÜ
72Ç                  ÜÜÜÜÜÜÜÜÜÜÜ                  ÜÜÜ
66Ç                  ÜÜÜÜÜÜÜÜÜÜÜ      Ü           ÜÜÜ
60Ç                 ÜÜÜÜÜÜÜÜÜÜÜÜ Ü  ÜÜÜÜ ÜÜ   Ü   ÜÜÜ
54Ç                 ÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜ  ÜÜÜÜ ÜÜÜÜÜ
48Ç                ÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜ ÜÜÜÜÜÜÜÜÜÜ
42Ç                ÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜ
36Ç                ÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜÜ

   0         383        766        1150       1533       1916       2300

                    HORIZONTAL DISTANCE ft
```

CELL DATA OUTPUT

WHR.DAT

REMARKS: SOUTH WILD HORSE RIDGE, B-B'

DATA COLLECTED AT END OF EACH CELL

CELL #	MAXIMUM VELOCITY (ft/sec)	AVERAGE VELOCITY	STANDARD DEVIATION VELOCITY	AVERAGE BOUNCE HEIGHT (ft)	MAXIMUM BOUNCE HEIGHT (ft)
1	NO ROCKS PASSED POINT				
2	NO ROCKS PASSED POINT				
3	NO ROCKS PASSED POINT				
4	NO ROCKS PASSED POINT				
5	NO ROCKS PASSED POINT				
6	90	64	19.38	4	14
7	72	39	15.02	5	20
8	42	25	11.47	3	6
9	37	22	0.00	2	6
10	46	37	0.00	3	6
11	62	55	0.00	58	74
12	28	28	0.00	7	7
13	NO ROCKS PASSED POINT				

X INTERVAL	ROCKS STOPPED
0 ft TO 10 ft	1
910 ft TO 920 ft	1
970 ft TO 980 ft	1
1010 ft TO 1020 ft	1

1020 ft	TO 1030 ft	1
1050 ft	TO 1060 ft	1
1090 ft	TO 1100 ft	2
1100 ft	TO 1110 ft	1
1120 ft	TO 1130 ft	1
1130 ft	TO 1140 ft	1
1150 ft	TO 1160 ft	3
1170 ft	TO 1180 ft	1
1200 ft	TO 1210 ft	1
1240 ft	TO 1250 ft	1
1270 ft	TO 1280 ft	1
1300 ft	TO 1310 ft	1
1390 ft	TO 1400 ft	1
1450 ft	TO 1460 ft	1
1470 ft	TO 1480 ft	1
1850 ft	TO 1860 ft	1
1970 ft	TO 1980 ft	1
2040 ft	TO 2050 ft	1

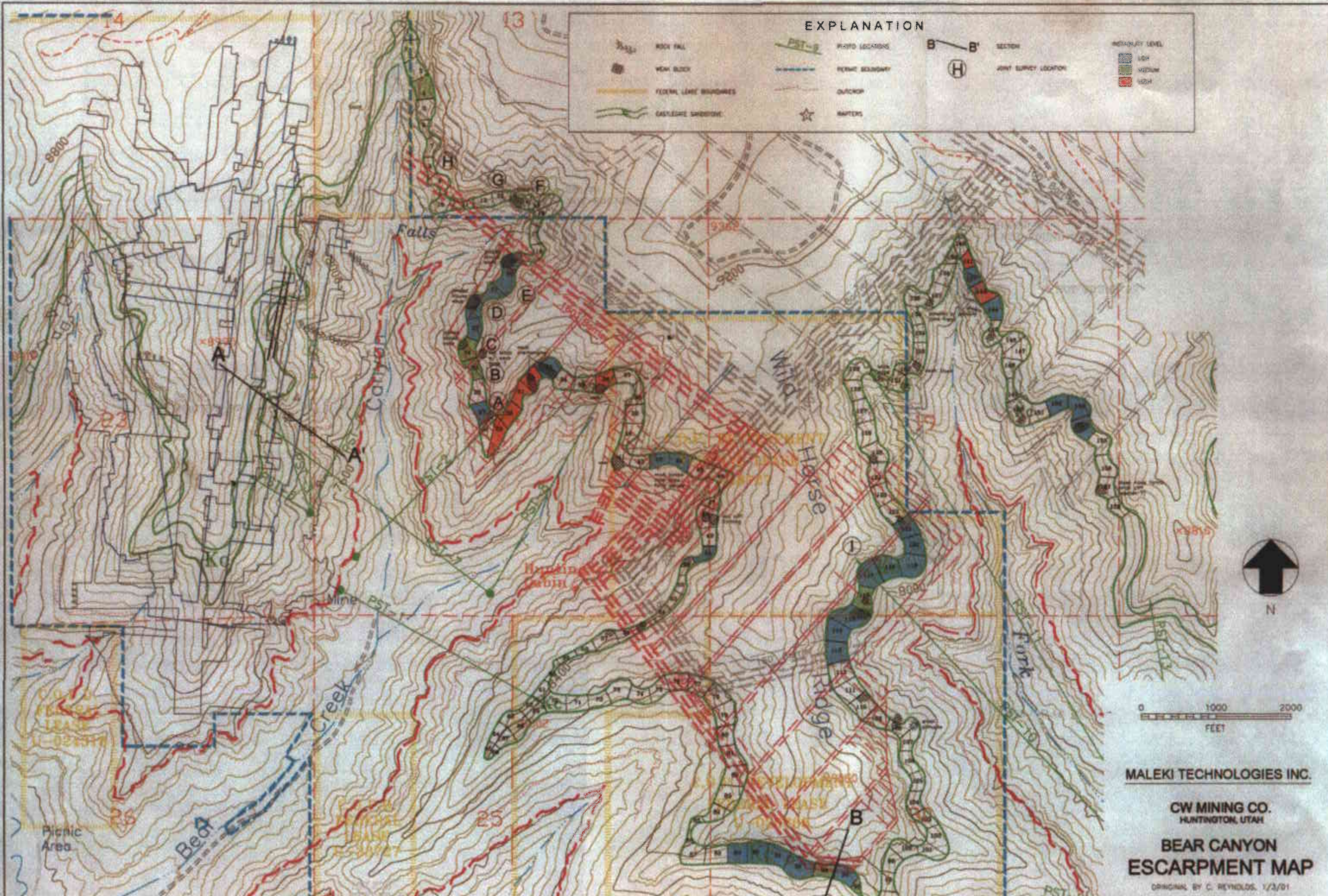


Figure 2. Escarpment modeling study.

CRSP Input File - C:\Program Files\Crsp\A-A'.DAT

Input File Specifications

Units of Measure: U.S.
Total Number of Cells: 9
Analysis Point X-Coordinate 1: 650
Analysis Point X-Coordinate 2: 0
Analysis Point X-Coordinate 3: 0
Initial Y-Top Starting Zone Coordinate: 8560
Initial Y-Base Starting Zone Coordinate: 8400
Remarks:

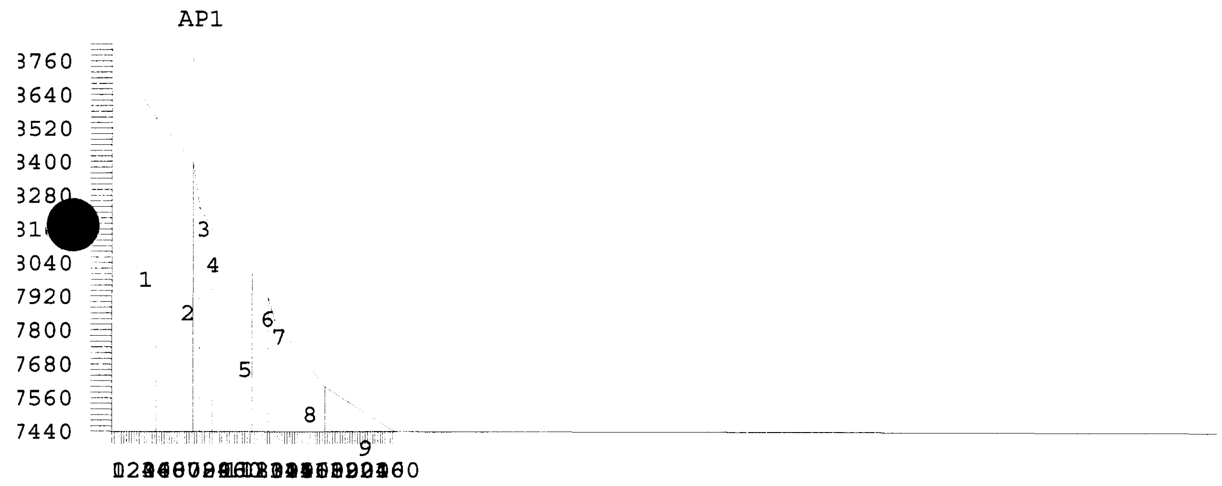
Cell Data

Cell No.	Surface R.	Tangent C.	Normal C.	Begin X	Begin Y	End X	End Y
1	5	.85	.35	0	8800	350	8560
2	1	.87	.37	350	8560	650	8400
3	5	.85	.35	650	8400	700	8240
4	5	.87	.35	700	8240	800	8160
5	5	.82	.33	800	8160	1120	8000
6	5	.82	.33	1120	8000	1250	7920
7	5	.82	.33	1250	7920	1300	7840
8	5	.82	.33	1300	7840	1700	7600
9	5	.82	.33	1700	7600	2250	7440

C:\Program Files\Crsp\A-A'.DAT   Total Rocks Rolled: 25

Spherical Rock: 6-ft dia., 18661-lb

Scale: Each division = 20 feet



CRSP Analysis Point Data - C:\Program Files\Crsp\A-A'.DAT

Analysis Point 1

Analysis Point 1: X = 650, Y = 8400

Spherical Rock: 6-ft dia., 18661-lb

Total Rocks Passing Analysis Point: 24

<u>Velocity (ft/sec)</u>

Maximum: 64.28
Average: 45.19
Minimum: 16.2
Std. Dev.: 13.72

<u>Bounce Height (ft)</u>

Maximum: 4.63
Average: 1.84
G. Mean: .76
Std. Dev.: 9.73

<u>Kinetic Energy (ft-lb)</u>

Maximum: 1473010
Average: 837192
Std. Dev.: 434819

CRSP Analysis Point Statistical Analysis - C:\Program Files\Crsp\A-A'.DAT

Analysis Point 1

Analysis Point 1: X = 650, Y = 8400

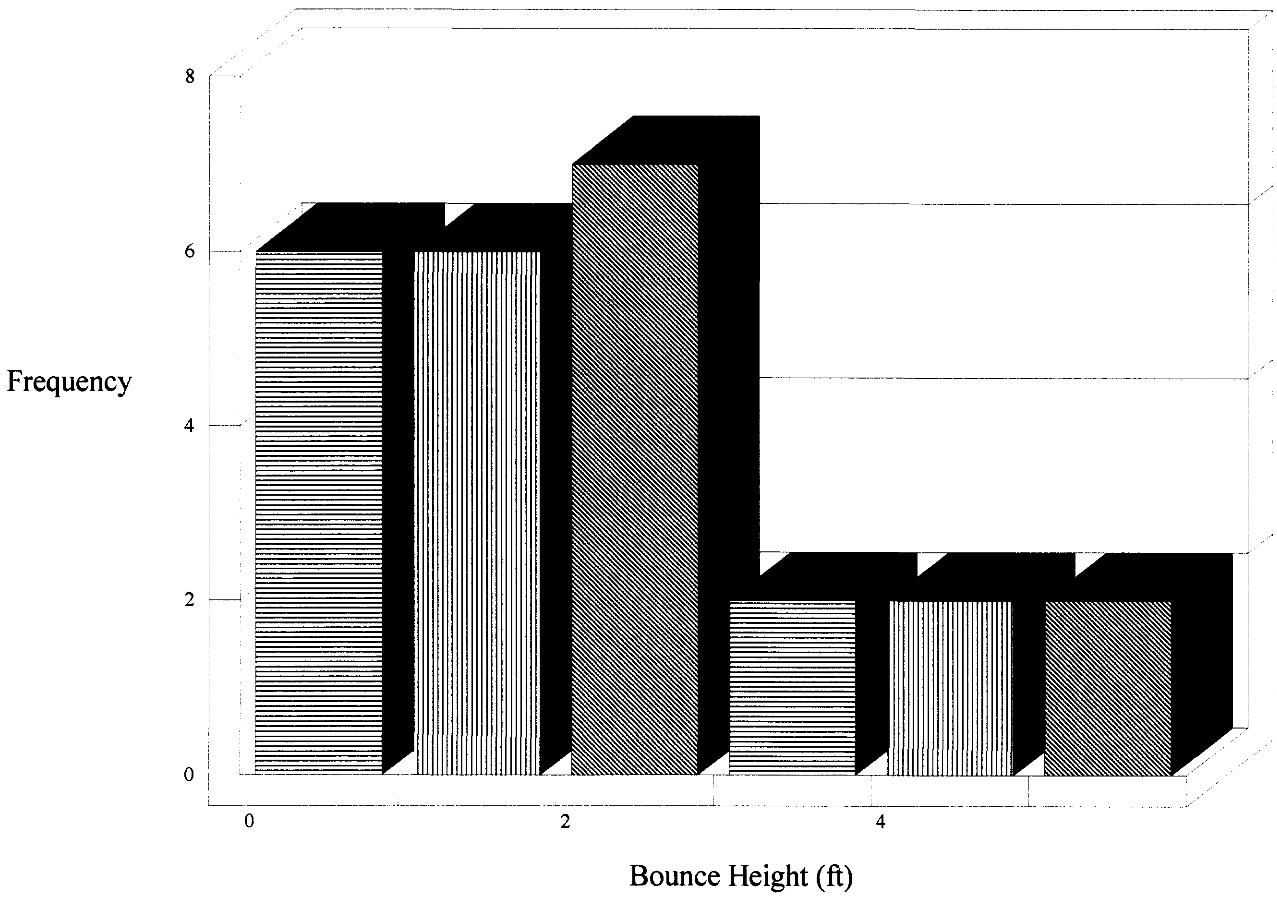
Spherical Rock: 6-ft dia., 18661-lb

Total Rocks Passing Analysis Point: 24

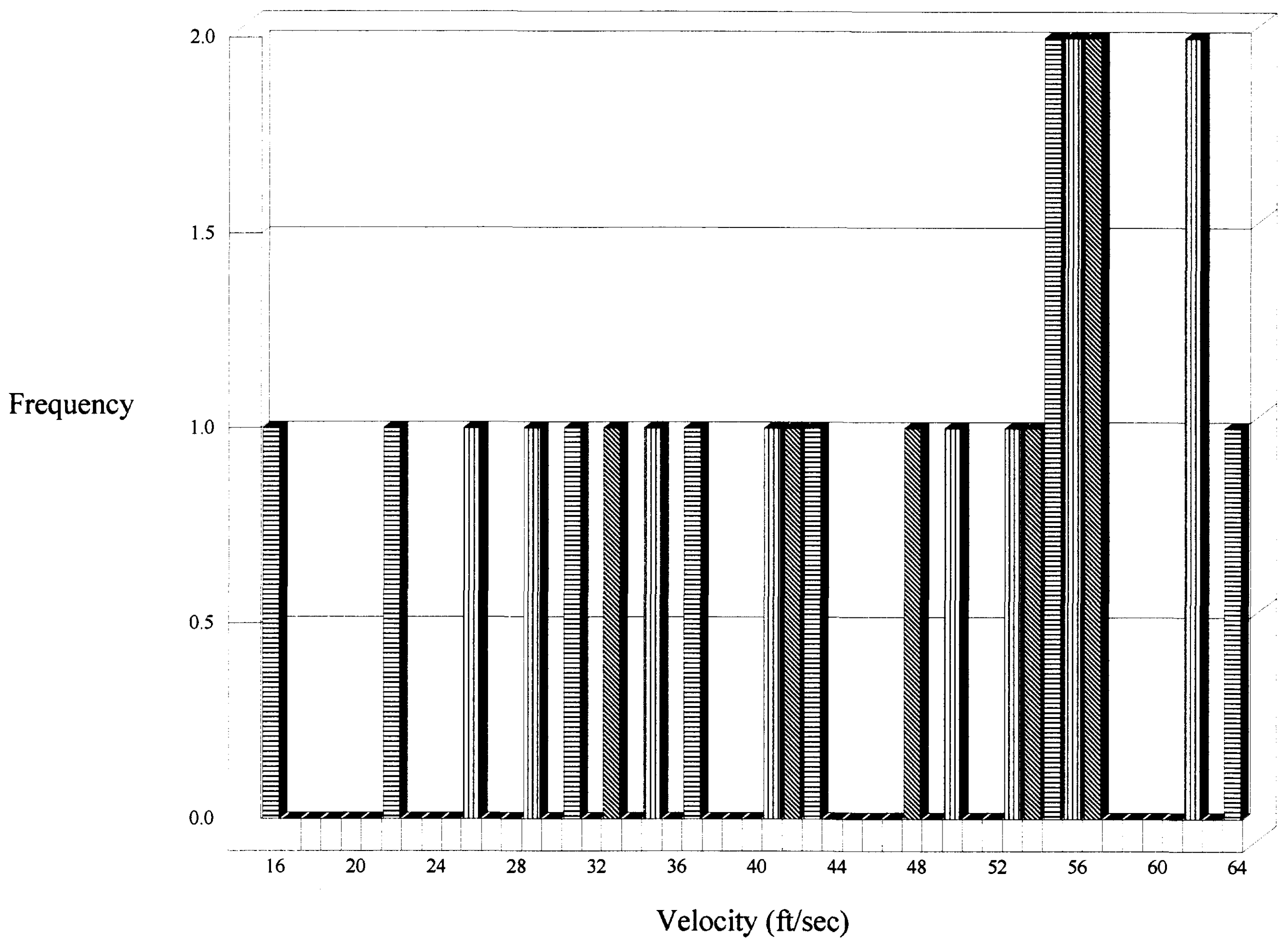
Cumulative Probability	Velocity (ft/sec)	Energy (ft-lb)	Bounce Height (ft)
50%	45.19	837192	0.76
75%	54.45	1130782	7.33
90%	62.78	1394848	13.24
95%	67.79	1553383	16.79
98%	73.4	1731311	20.78

Note: Velocity and kinetic energy are analyzed assuming a normal distribution.
Bounce height is analyzed assuming a log distribution.

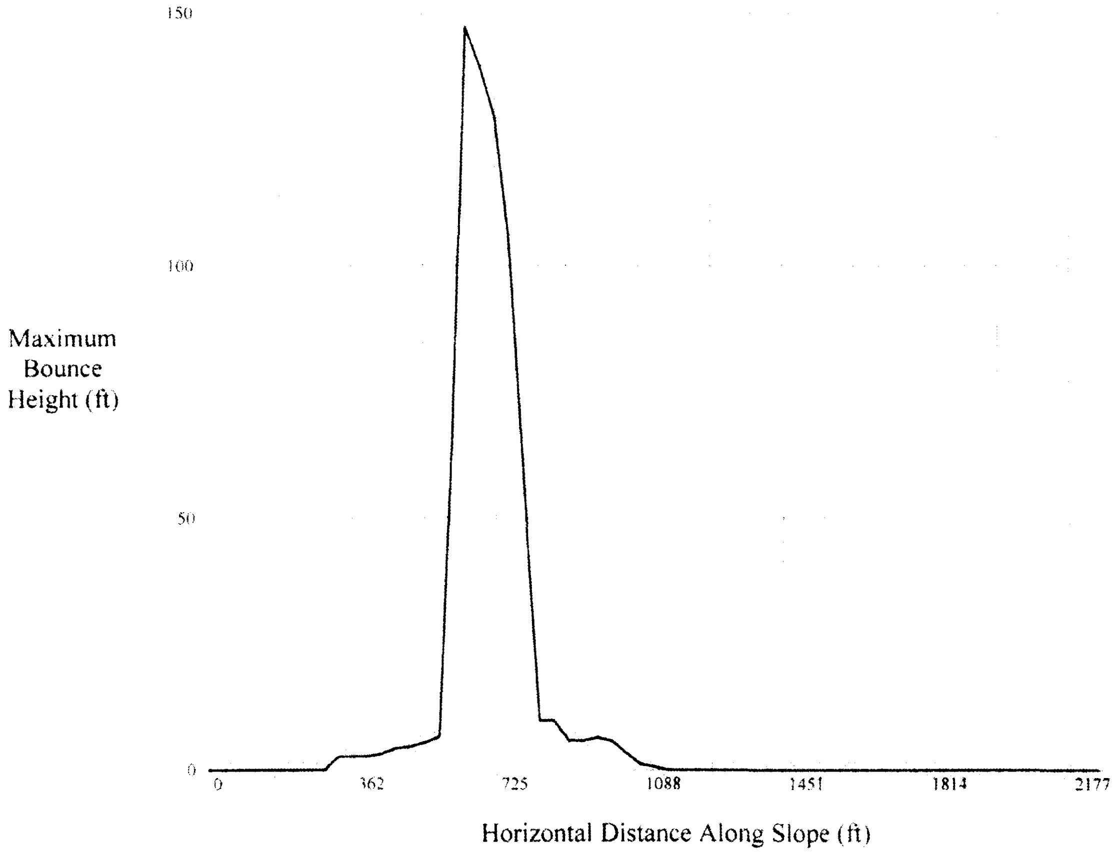
Analysis Point 1 Bounce Height Distribution - C:\Program Files\Crsp\A-A'.DAT

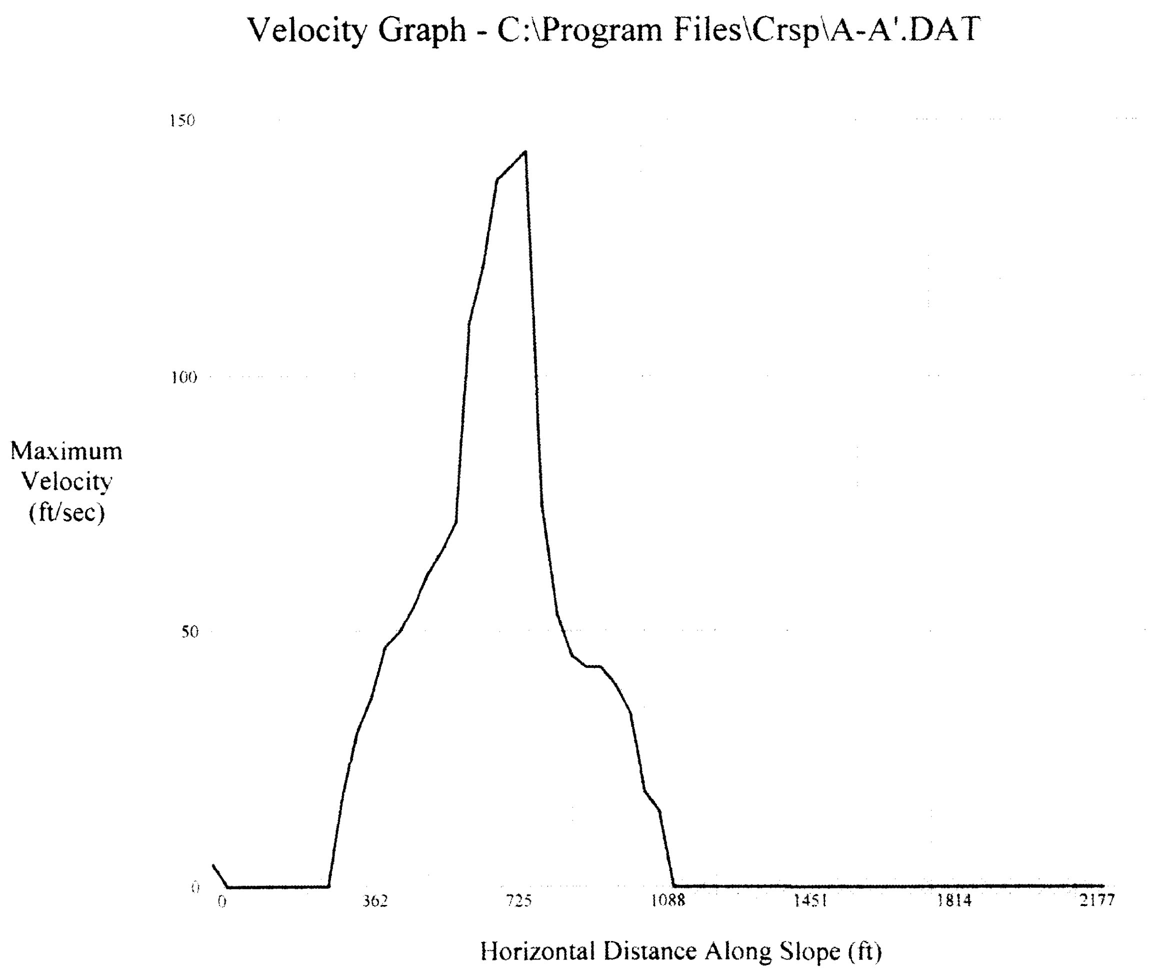


Analysis Point 1 Velocity Distribution - C:\Program Files\Crsp\A-A'.DAT



Bounce Height Graph - C:\Program Files\Crsp\A-A'.DAT




Velocity Graph - C:\Program Files\Crsp\A-A'.DAT
150
100
50
0
Maximum Velocity (ft/sec)
0
362
725
1088
1451
1814
2177
Horizontal Distance Along Slope (ft)

CRSP Data Collected at End of Each Cell - C:\Program Files\Crsp\A-A'.DAT

Velocity Units: ft/sec   Bounce Height Units: ft

Cell No.	Max. Velocity	Avg. Velocity	Std. Dev. Velocity	Max. Bounce Ht.	Avg. Bounce Ht.
1	No rocks	past end of cell			
2	62	44	13.69	6	1
3	96	79	9.38	128	92
4	135	58	41.56	50	9
5	39	25	0	2	1
6	37	26	0	4	2
7	68	59	0	33	22
8	27	27	0	2	2
9	No rocks	past end of cell			

CRSP Rocks Stopped Data - C:\Program Files\Crsp\A-A'.DAT

X Interval	Rocks Stopped
0 To 10 ft	1
10 To 20 ft	0
20 To 30 ft	0
30 To 40 ft	0
40 To 50 ft	0
50 To 60 ft	0
60 To 70 ft	0
70 To 80 ft	0
80 To 90 ft	0
90 To 100 ft	0
100 To 110 ft	0
110 To 120 ft	0
120 To 130 ft	0
130 To 140 ft	0
140 To 150 ft	0
150 To 160 ft	0
160 To 170 ft	0
170 To 180 ft	0
180 To 190 ft	0
190 To 200 ft	0
200 To 210 ft	0
210 To 220 ft	0
220 To 230 ft	0
230 To 240 ft	0
240 To 250 ft	0
250 To 260 ft	0
260 To 270 ft	0
270 To 280 ft	0
280 To 290 ft	0
290 To 300 ft	0
300 To 310 ft	0
310 To 320 ft	0
320 To 330 ft	0
330 To 340 ft	0
340 To 350 ft	0
350 To 360 ft	0
360 To 370 ft	0
370 To 380 ft	0
380 To 390 ft	0
390 To 400 ft	0
400 To 410 ft	0
410 To 420 ft	0
420 To 430 ft	0
430 To 440 ft	0

X Interval	Rocks Stopped
440 To 450 ft	0
450 To 460 ft	0
460 To 470 ft	0
470 To 480 ft	0
480 To 490 ft	0
490 To 500 ft	0
500 To 510 ft	0
510 To 520 ft	0
520 To 530 ft	0
530 To 540 ft	0
540 To 550 ft	0
550 To 560 ft	0
560 To 570 ft	0
570 To 580 ft	0
580 To 590 ft	0
590 To 600 ft	0
600 To 610 ft	0
610 To 620 ft	0
620 To 630 ft	0
630 To 640 ft	0
640 To 650 ft	0
650 To 660 ft	0
660 To 670 ft	0
670 To 680 ft	0
680 To 690 ft	0
690 To 700 ft	0
700 To 710 ft	0
710 To 720 ft	0
720 To 730 ft	0
730 To 740 ft	0
740 To 750 ft	0
750 To 760 ft	1
760 To 770 ft	0
770 To 780 ft	0
780 To 790 ft	0
790 To 800 ft	0
800 To 810 ft	2
810 To 820 ft	0
820 To 830 ft	4
830 To 840 ft	0
840 To 850 ft	1
850 To 860 ft	0
860 To 870 ft	0
870 To 880 ft	1
880 To 890 ft	1

X Interval	Rocks Stopped
890 To 900 ft	0
900 To 910 ft	3
910 To 920 ft	1
920 To 930 ft	0
930 To 940 ft	2
940 To 950 ft	1
950 To 960 ft	1
960 To 970 ft	0
970 To 980 ft	1
980 To 990 ft	0
990 To 1000 ft	1
1000 To 1010 ft	1
1010 To 1020 ft	0
1020 To 1030 ft	0
1030 To 1040 ft	0
1040 To 1050 ft	0
1050 To 1060 ft	0
1060 To 1070 ft	0
1070 To 1080 ft	0
1080 To 1090 ft	0
1090 To 1100 ft	0
1100 To 1110 ft	0
1110 To 1120 ft	0
1120 To 1130 ft	1
1130 To 1140 ft	0
1140 To 1150 ft	0
1150 To 1160 ft	0
1160 To 1170 ft	0
1170 To 1180 ft	0
1180 To 1190 ft	0
1190 To 1200 ft	0
1200 To 1210 ft	0
1210 To 1220 ft	0
1220 To 1230 ft	0
1230 To 1240 ft	0
1240 To 1250 ft	0
1250 To 1260 ft	0
1260 To 1270 ft	0
1270 To 1280 ft	0
1280 To 1290 ft	0
1290 To 1300 ft	0
1300 To 1310 ft	0
1310 To 1320 ft	0
1320 To 1330 ft	0
1330 To 1340 ft	0
1340 To 1350 ft	0
1350 To 1360 ft	0
1360 To 1370 ft	0
1370 To 1380 ft	0

CRSP Input File - C:\Program Files\Crsp\B-B'.dat

Input File Specifications

Units of Measure: U.S.
Total Number of Cells: 13
Analysis Point X-Coordinate 1: 800
Analysis Point X-Coordinate 2:
Analysis Point X-Coordinate 3:
Initial Y-Top Starting Zone Coordinate: 8400
Initial Y-Base Starting Zone Coordinate: 8160
Remarks:

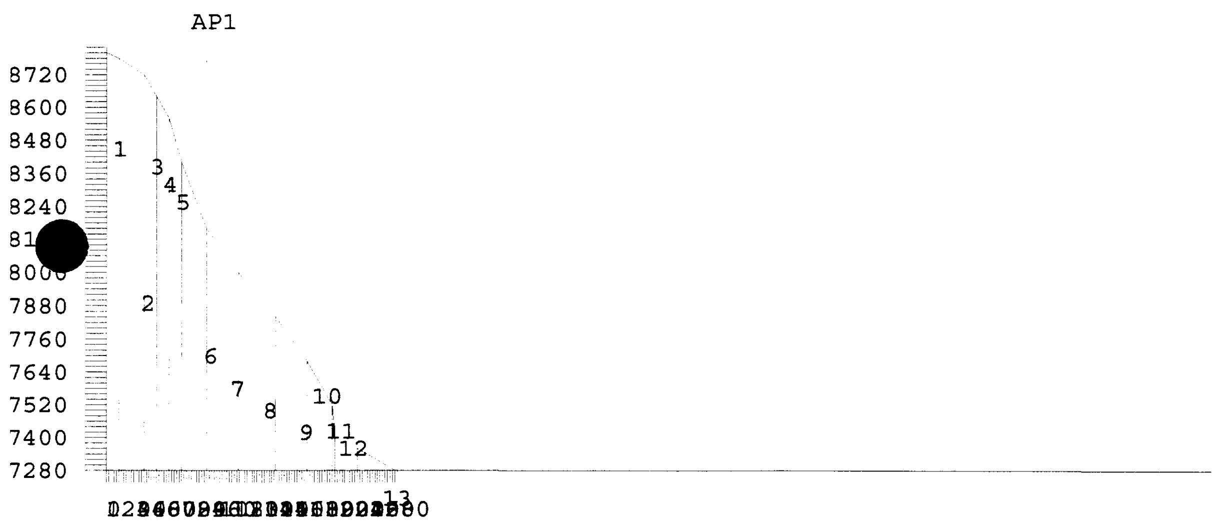
Cell Data

Cell No.	Surface R.	Tangent C.	Normal C.	Begin X	Begin Y	End X	End Y
1	5	.85	.35	0	8800	100	8780
2	5	.85	.35	100	8780	300	8720
3	5	.85	.35	300	8720	400	8640
4	5	.85	.35	400	8640	500	8560
5	5	.85	.35	500	8560	600	8400
6	1	.87	.37	600	8400	800	8160
7	5	.85	.35	800	8160	1050	8000
8	5	.87	.35	1050	8000	1350	7840
9	5	.82	.33	1350	7840	1600	7680
10	5	.82	.33	1600	7680	1800	7520
11	5	.82	.33	1800	7520	1820	7440
12	5	.82	.33	1820	7440	2000	7360
13	5	.82	.33	2000	7360	2300	7280

C:\Program Files\Crsp\B-B'.dat   Total Rocks Rolled: 25

Spherical Rock: 6-ft dia., 18661-lb

Scale: Each division = 20 feet



CRSP Analysis Point Data - C:\Program Files\Crsp\B-B'.dat

Analysis Point 1

Analysis Point 1: X = 800, Y = 8160

Spherical Rock: 6-ft dia., 18661-lb

Total Rocks Passing Analysis Point: 24

Velocity (ft/sec)	Bounce Height (ft)	Kinetic Energy (ft-lb)
Maximum: 87.59	Maximum: 18.99	Maximum: 2770052
Average: 61.96	Average: 6.35	Average: 1512508
Minimum: 23.02	G. Mean: 2.79	Std. Dev.: 760375
Std. Dev.: 18.65	Std. Dev.: 8.54	

CRSP Analysis Point Statistical Analysis - C:\Program Files\Crsp\B-B'.dat

Analysis Point 1

Analysis Point 1: X = 800, Y = 8160

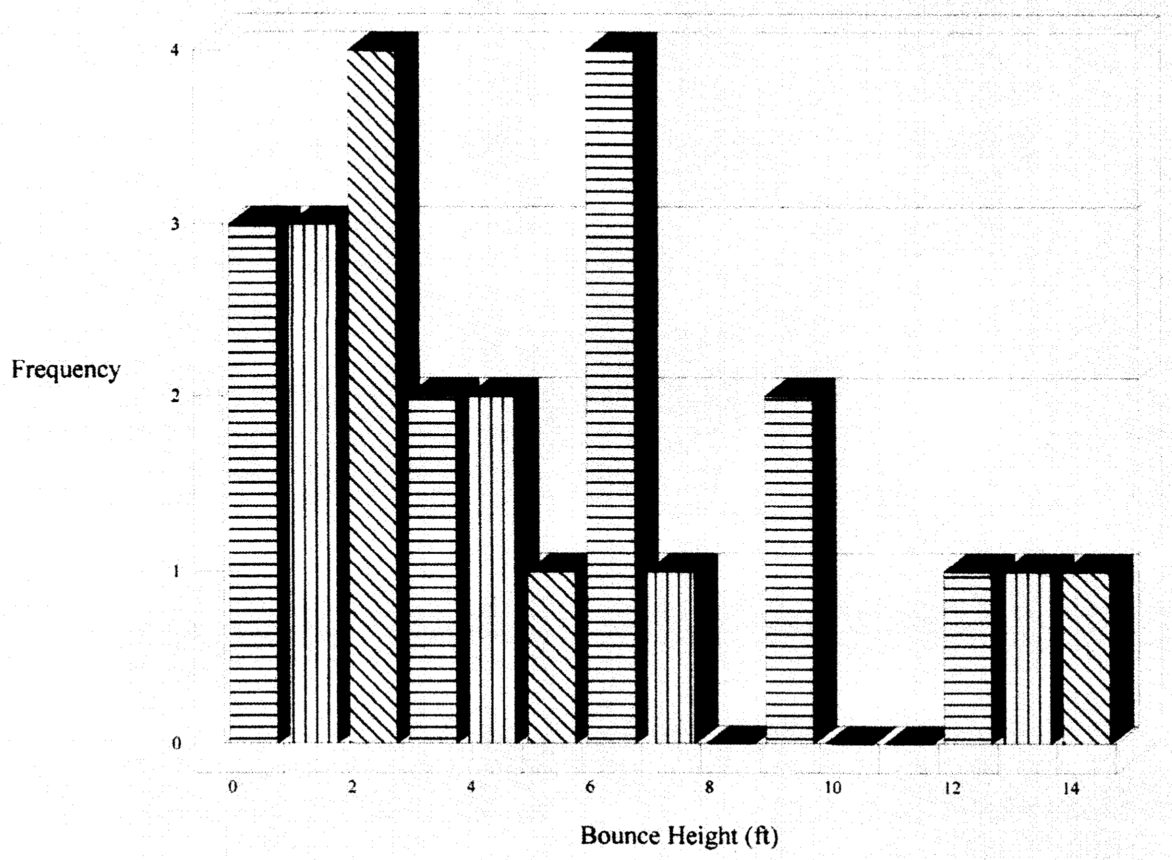
Spherical Rock: 6-ft dia., 18661-lb

Total Rocks Passing Analysis Point: 24

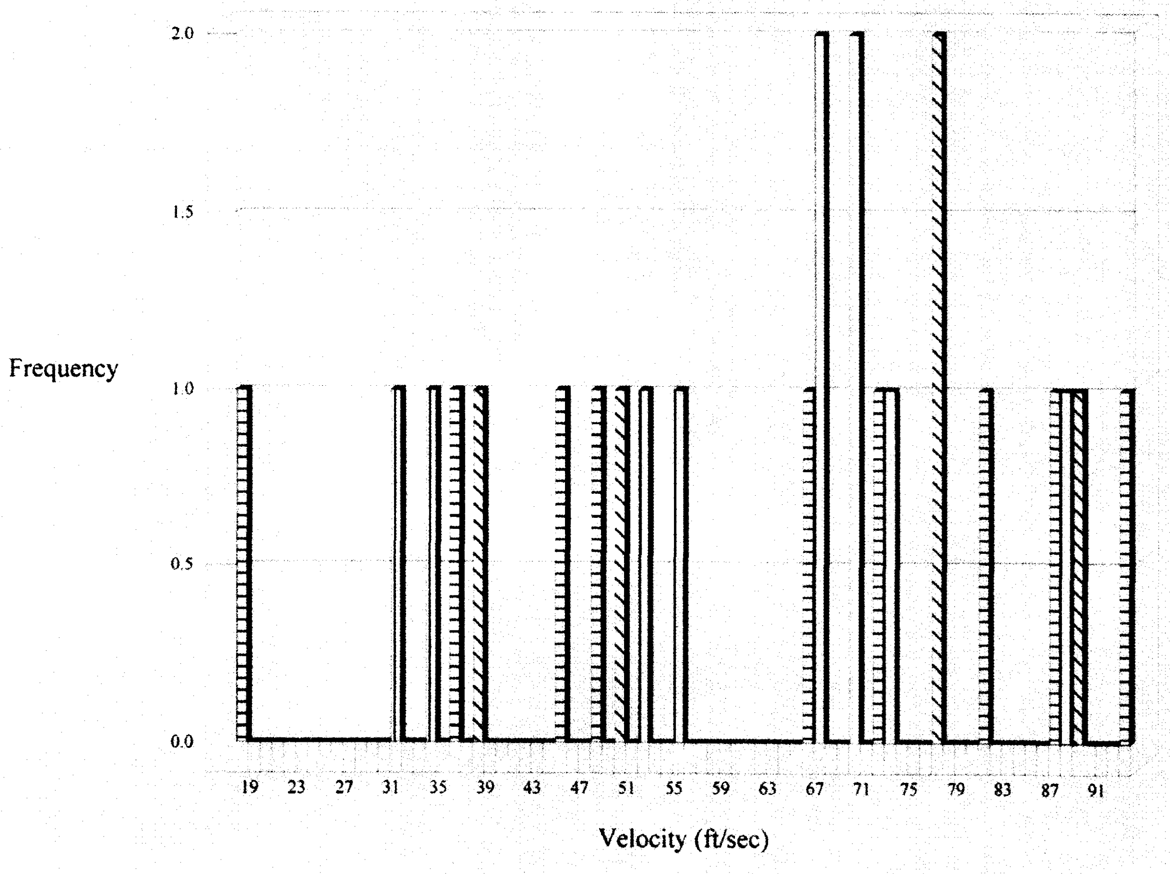
Cumulative Probability	Velocity (ft/sec)	Energy (ft-lb)	Bounce Height (ft)
50%	61.96	1512508	2.79
75%	74.55	2025914	8.55
90%	85.88	2487690	13.74
95%	92.68	2764923	16.85
98%	100.32	3076069	20.35

Note: Velocity and kinetic energy are analyzed assuming a normal distribution.
Bounce height is analyzed assuming a log distribution.

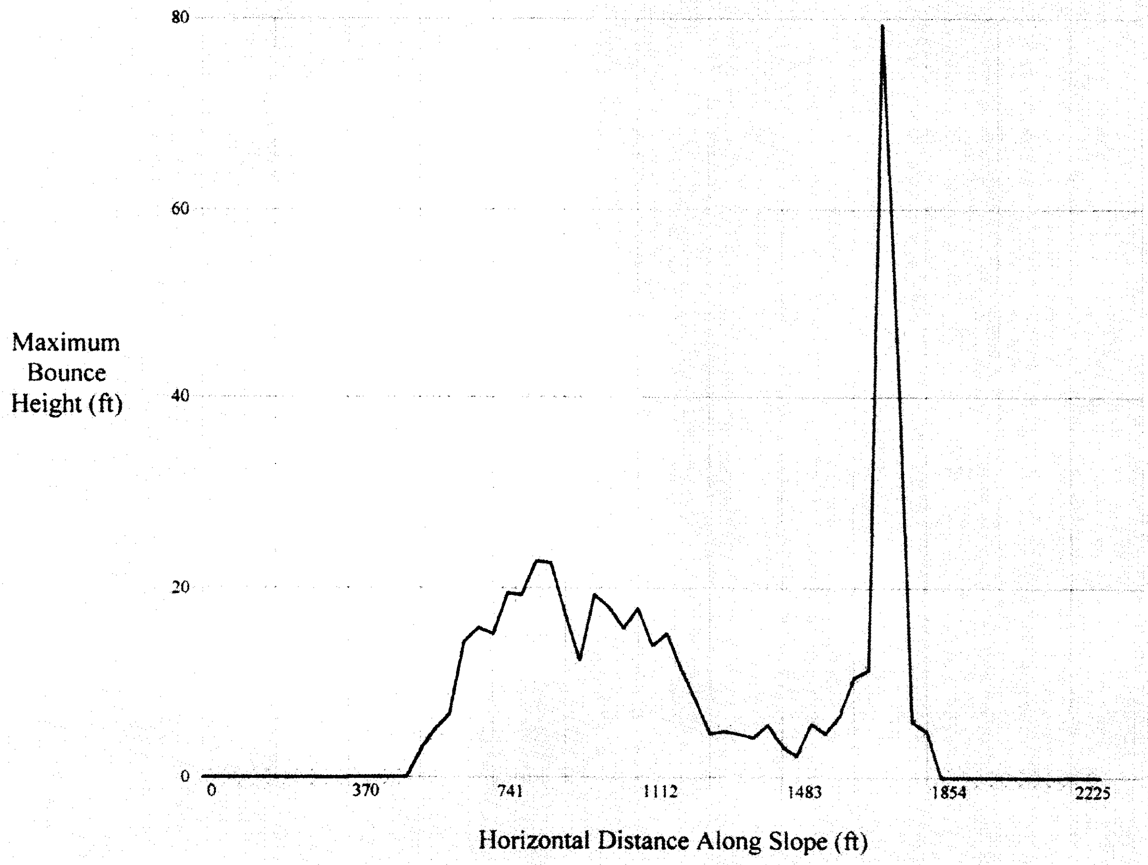
Analysis Point 1 Bounce Height Distribution - C:\Program Files\Crsp\B-B'.dat



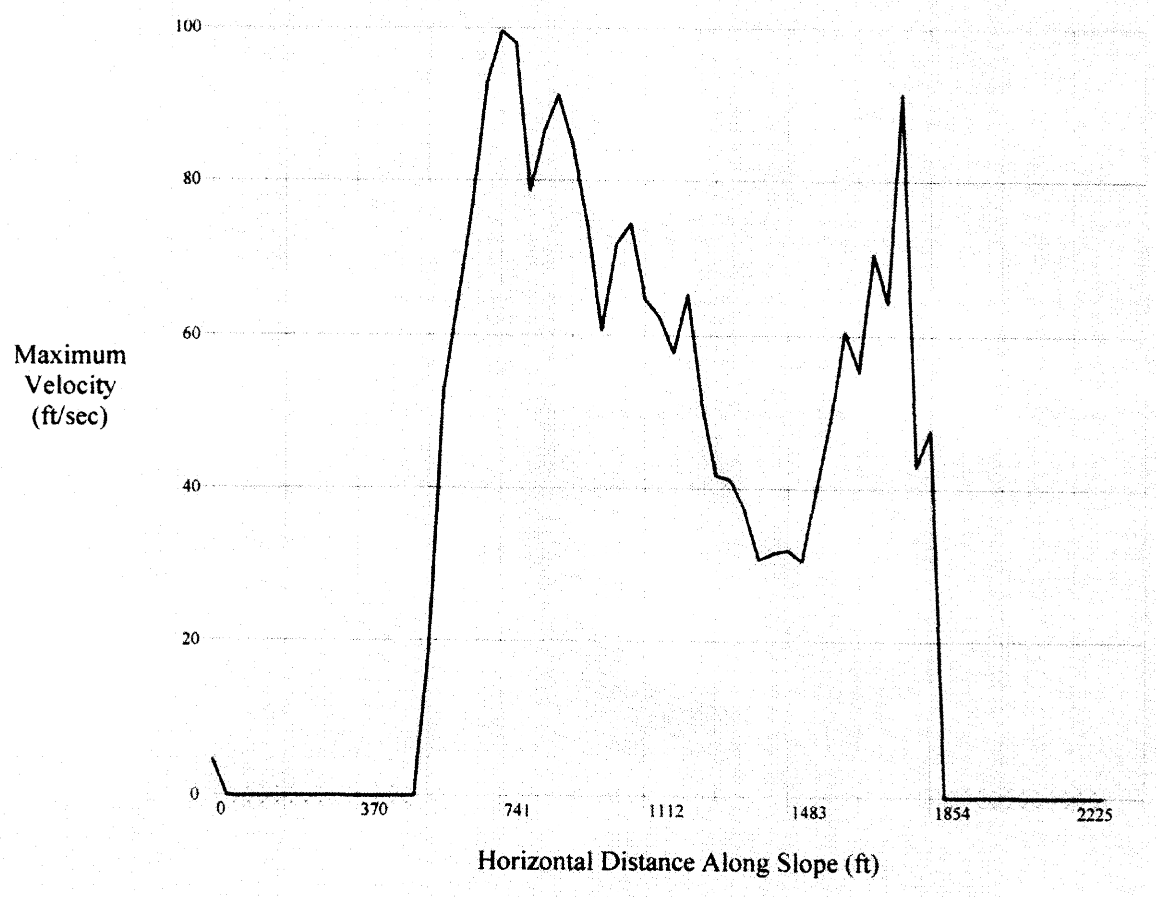
Analysis Point 1 Velocity Distribution - C:\Program Files\Crsp\B-B'.dat



Bounce Height Graph - C:\Program Files\Crsp\B-B'.dat



Velocity Graph - C:\Program Files\Crsp\B-B'.dat



CRSP Data Collected at End of Each Cell - C:\Program Files\Crsp\B-B'.dat

Velocity Units: ft/sec   Bounce Height Units: ft

Cell No.	Max. Velocity	Avg. Velocity	Std. Dev. Velocity	Max. Bounce Ht.	Avg. Bounce Ht.
1	No rocks	past end of cell			
2	No rocks	past end of cell			
3	No rocks	past end of cell			
4	No rocks	past end of cell			
5	No rocks	past end of cell			
6	94	63	20.74	14	4
7	52	32	13.15	18	3
8	18	16	0	4	1
9	21	21	0	1	0
10	31	31	0	12	11
11	46	46	0	74	74
12	No rocks	past end of cell			
13	No rocks	past end of cell			

CRSP Rocks Stopped Data - C:\Program Files\Crsp\B-B'.dat

X Interval	Rocks Stopped
0 To 10 ft	1
10 To 20 ft	0
20 To 30 ft	0
30 To 40 ft	0
40 To 50 ft	0
50 To 60 ft	0
60 To 70 ft	0
70 To 80 ft	0
80 To 90 ft	0
90 To 100 ft	0
100 To 110 ft	0
110 To 120 ft	0
120 To 130 ft	0
130 To 140 ft	0
140 To 150 ft	0
150 To 160 ft	0
160 To 170 ft	0
170 To 180 ft	0
180 To 190 ft	0
190 To 200 ft	0
200 To 210 ft	0
210 To 220 ft	0
220 To 230 ft	0
230 To 240 ft	0
240 To 250 ft	0
250 To 260 ft	0
260 To 270 ft	0
270 To 280 ft	0
280 To 290 ft	0
290 To 300 ft	0
300 To 310 ft	0
310 To 320 ft	0
320 To 330 ft	0
330 To 340 ft	0
340 To 350 ft	0
350 To 360 ft	0
360 To 370 ft	0
370 To 380 ft	0
380 To 390 ft	0
390 To 400 ft	0
400 To 410 ft	0
410 To 420 ft	0
420 To 430 ft	0
430 To 440 ft	0

X Interval	Rocks Stopped
440 To 450 ft	0
450 To 460 ft	0
460 To 470 ft	0
470 To 480 ft	0
480 To 490 ft	0
490 To 500 ft	0
500 To 510 ft	0
510 To 520 ft	0
520 To 530 ft	0
530 To 540 ft	0
540 To 550 ft	0
550 To 560 ft	0
560 To 570 ft	0
570 To 580 ft	0
580 To 590 ft	0
590 To 600 ft	0
600 To 610 ft	0
610 To 620 ft	0
620 To 630 ft	0
630 To 640 ft	0
640 To 650 ft	0
650 To 660 ft	0
660 To 670 ft	0
670 To 680 ft	0
680 To 690 ft	0
690 To 700 ft	0
700 To 710 ft	0
710 To 720 ft	0
720 To 730 ft	0
730 To 740 ft	0
740 To 750 ft	0
750 To 760 ft	0
760 To 770 ft	0
770 To 780 ft	0
780 To 790 ft	0
790 To 800 ft	0
800 To 810 ft	0
810 To 820 ft	0
820 To 830 ft	0
830 To 840 ft	0
840 To 850 ft	0
850 To 860 ft	0
860 To 870 ft	0
870 To 880 ft	0
880 To 890 ft	0

X Interval	Rocks Stopped
890 To 900 ft	0
900 To 910 ft	0
910 To 920 ft	1
920 To 930 ft	1
930 To 940 ft	0
940 To 950 ft	0
950 To 960 ft	0
960 To 970 ft	1
970 To 980 ft	0
980 To 990 ft	1
990 To 1000 ft	0
1000 To 1010 ft	0
1010 To 1020 ft	1
1020 To 1030 ft	0
1030 To 1040 ft	1
1040 To 1050 ft	0
1050 To 1060 ft	2
1060 To 1070 ft	0
1070 To 1080 ft	1
1080 To 1090 ft	0
1090 To 1100 ft	0
1100 To 1110 ft	2
1110 To 1120 ft	0
1120 To 1130 ft	2
1130 To 1140 ft	0
1140 To 1150 ft	0
1150 To 1160 ft	0
1160 To 1170 ft	2
1170 To 1180 ft	1
1180 To 1190 ft	0
1190 To 1200 ft	0
1200 To 1210 ft	1
1210 To 1220 ft	0
1220 To 1230 ft	0
1230 To 1240 ft	0
1240 To 1250 ft	0
1250 To 1260 ft	0
1260 To 1270 ft	0
1270 To 1280 ft	1
1280 To 1290 ft	2
1290 To 1300 ft	1
1300 To 1310 ft	0
1310 To 1320 ft	1
1320 To 1330 ft	0
1330 To 1340 ft	0
1340 To 1350 ft	0
1350 To 1360 ft	0
1360 To 1370 ft	0
1370 To 1380 ft	0

CRSP Input File - C:\Program Files\Crsp\C-C'.dat

Input File Specifications

Units of Measure: U.S.
Total Number of Cells: 9
Analysis Point X-Coordinate 1: 200
Analysis Point X-Coordinate 2:
Analysis Point X-Coordinate 3:
Initial Y-Top Starting Zone Coordinate: 8560
Initial Y-Base Starting Zone Coordinate: 8400
Remarks:

Cell Data

Cell No.	Surface R.	Tangent C.	Normal C.	Begin X	Begin Y	End X	End Y
1	5	.85	.35	0	8650	100	8560
2	1	.87	.37	100	8560	200	8400
3	5	.85	.35	200	8400	300	8320
4	5	.87	.35	300	8320	500	8160
5	5	.82	.33	500	8160	650	8080
6	5	.82	.33	650	8080	850	7920
7	5	.82	.33	850	7920	1050	7760
8	5	.82	.33	1050	7760	1480	7600
9	5	.82	.33	1480	7600	1530	7520

C:\Program Files\Crsp\C-C'.dat   Total Rocks Rolled: 25

Spherical Rock: 6-ft dia., 18661-lb

Scale: Each division = 20 feet

CRSP Analysis Point Data - C:\Program Files\Crsp\C-C'.dat

Analysis Point 1

Analysis Point 1: X = 200, Y = 8400

Spherical Rock: 6-ft dia., 18661-lb

Total Rocks Passing Analysis Point: 24

Velocity (ft/sec)

Maximum: 79.82
Average: 54.71
Minimum: 21.74
Std. Dev.: 15.41

Bounce Height (ft)

Maximum: 17.42
Average: 4.69
G. Mean: 1.89
Std. Dev.: 7.36

Kinetic Energy (ft-lb)

Maximum: 2011921
Average: 1142518
Std. Dev.: 562001

CRSP Analysis Point Statistical Analysis - C:\Program Files\Crsp\C-C'.dat

Analysis Point 1

Analysis Point 1: X = 200, Y = 8400

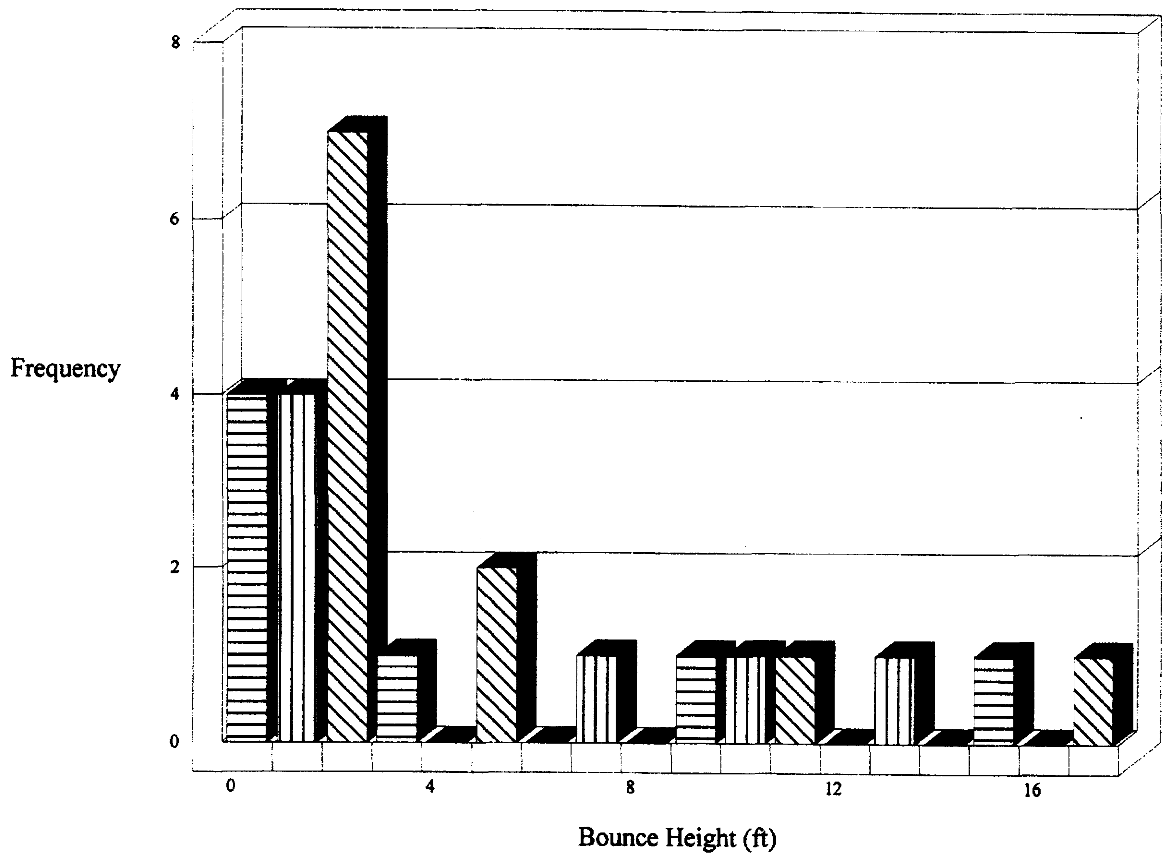
Spherical Rock: 6-ft dia., 18661-lb

Total Rocks Passing Analysis Point: 24

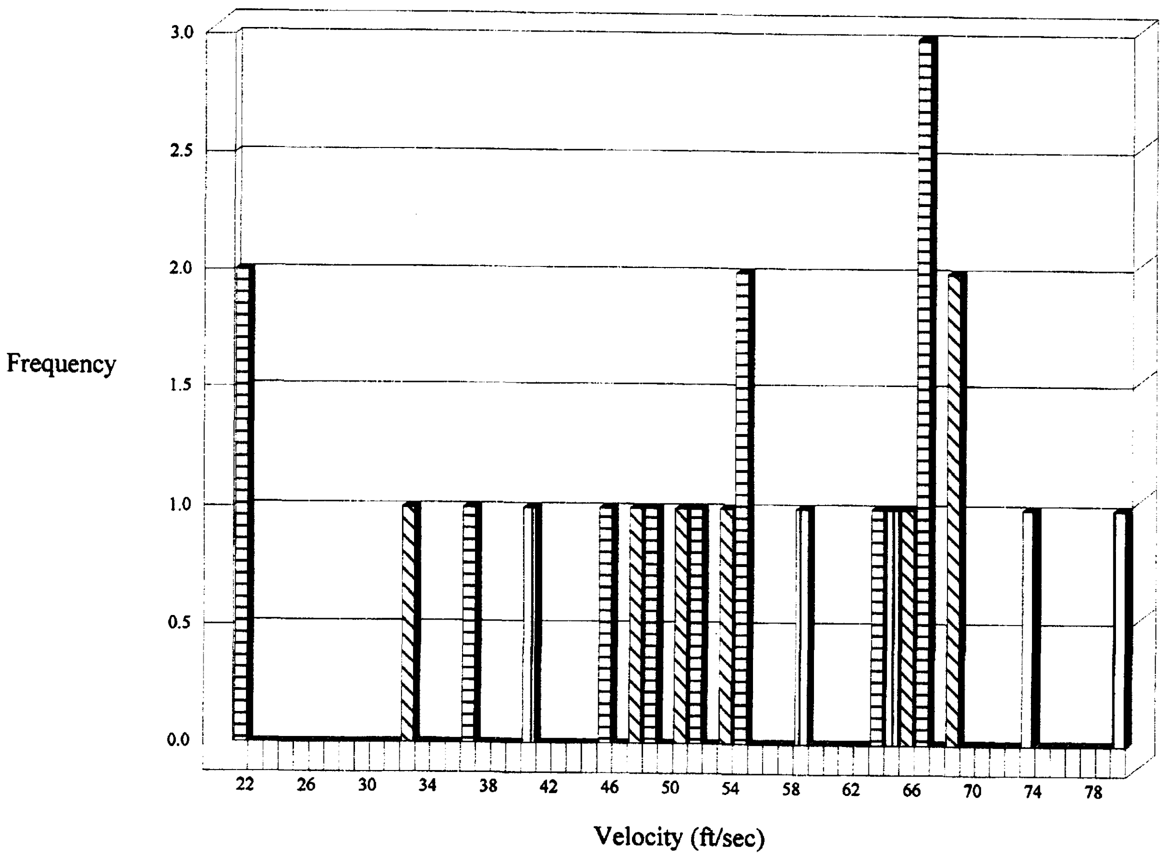
Cumulative Probability	Velocity (ft/sec)	Energy (ft-lb)	Bounce Height (ft)
50%	54.71	1142518	1.89
75%	65.11	1521981	6.86
90%	74.46	1863284	11.33
95%	80.08	2068190	14.01
98%	86.39	2298161	17.03

Note: Velocity and kinetic energy are analyzed assuming a normal distribution.
Bounce height is analyzed assuming a log distribution.

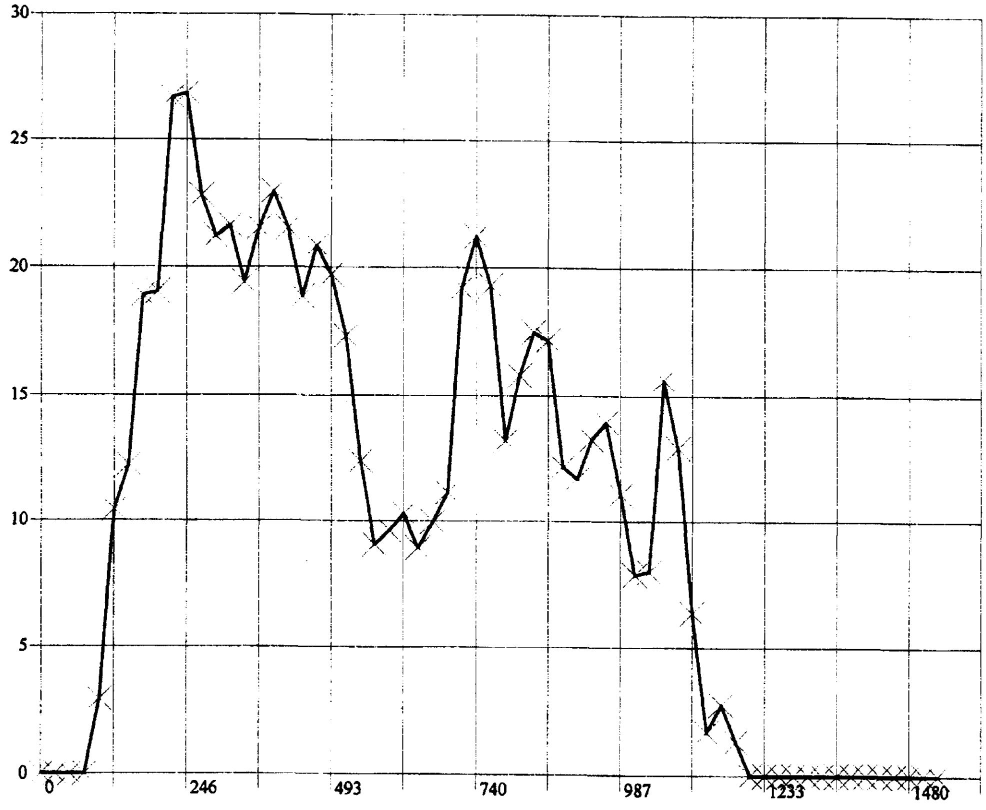
Analysis Point 1 Bounce Height Distribution - C:\Program Files\Crsp\C-C'.dat



Analysis Point 1 Velocity Distribution - C:\Program Files\Crsp\C-C'.dat



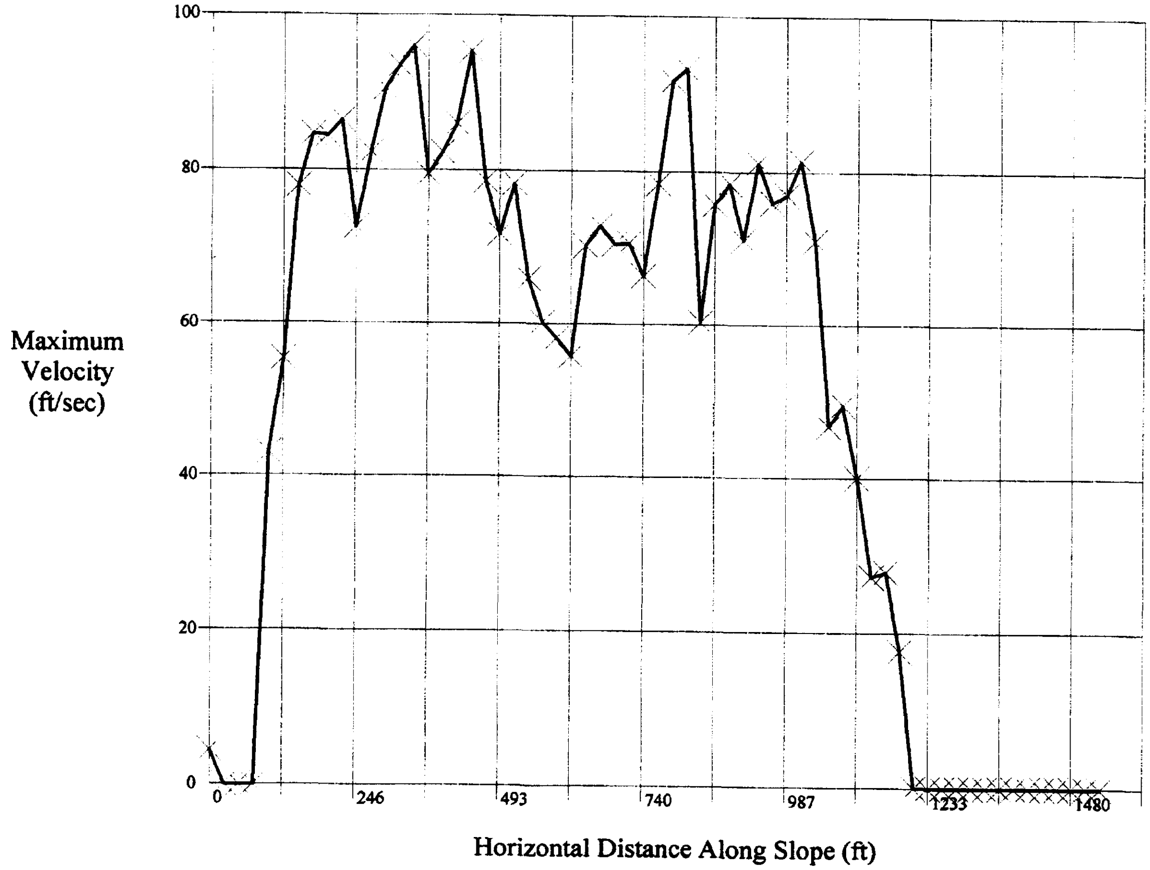
Bounce Height Graph - C:\Program Files\Crsp\C-C'.dat



Maximum Bounce Height (ft)

Horizontal Distance Along Slope (ft)

Velocity Graph - C:\Program Files\Crsp\C-C'.dat



CRSP Data Collected at End of Each Cell - C:\Program Files\Crsp\C-C'.dat

Velocity Units: ft/sec   Bounce Height Units: ft

Cell No.	Max. Velocity	Avg. Velocity	Std. Dev. Velocity	Max. Bounce Ht.	Avg. Bounce Ht.
1	No rocks	past end of cell			
2	80	55	15.41	17	4
3	80	45	18.4	16	5
4	63	43	13.95	20	5
5	50	32	12.48	10	3
6	88	45	19.81	11	3
7	79	39	19.54	8	3
8	No rocks	past end of cell			
9	No rocks	past end of cell			

CRSP Rocks Stopped Data - C:\Program Files\Crsp\C-C'.dat

X Interval	Rocks Stopped
0 To 10 ft	1
10 To 20 ft	0
20 To 30 ft	0
30 To 40 ft	0
40 To 50 ft	0
50 To 60 ft	0
60 To 70 ft	0
70 To 80 ft	0
80 To 90 ft	0
90 To 100 ft	0
100 To 110 ft	0
110 To 120 ft	0
120 To 130 ft	0
130 To 140 ft	0
140 To 150 ft	0
150 To 160 ft	0
160 To 170 ft	0
170 To 180 ft	0
180 To 190 ft	0
190 To 200 ft	0
200 To 210 ft	0
210 To 220 ft	0
220 To 230 ft	0
230 To 240 ft	0
240 To 250 ft	0
250 To 260 ft	0
260 To 270 ft	0
270 To 280 ft	0
280 To 290 ft	0
290 To 300 ft	0
300 To 310 ft	0
310 To 320 ft	0
320 To 330 ft	0
330 To 340 ft	0
340 To 350 ft	0
350 To 360 ft	0
360 To 370 ft	0
370 To 380 ft	0
380 To 390 ft	0
390 To 400 ft	0
400 To 410 ft	0
410 To 420 ft	0
420 To 430 ft	0
430 To 440 ft	0

X Interval	Rocks Stopped
440 To 450 ft	0
450 To 460 ft	0
460 To 470 ft	0
470 To 480 ft	0
480 To 490 ft	0
490 To 500 ft	0
500 To 510 ft	0
510 To 520 ft	3
520 To 530 ft	1
530 To 540 ft	0
540 To 550 ft	1
550 To 560 ft	1
560 To 570 ft	1
570 To 580 ft	0
580 To 590 ft	1
590 To 600 ft	1
600 To 610 ft	2
610 To 620 ft	0
620 To 630 ft	2
630 To 640 ft	1
640 To 650 ft	0
650 To 660 ft	0
660 To 670 ft	0
670 To 680 ft	0
680 To 690 ft	0
690 To 700 ft	0
700 To 710 ft	0
710 To 720 ft	0
720 To 730 ft	0
730 To 740 ft	0
740 To 750 ft	0
750 To 760 ft	0
760 To 770 ft	0
770 To 780 ft	0
780 To 790 ft	0
790 To 800 ft	0
800 To 810 ft	0
810 To 820 ft	0
820 To 830 ft	0
830 To 840 ft	0
840 To 850 ft	0
850 To 860 ft	0
860 To 870 ft	0
870 To 880 ft	0
880 To 890 ft	1

X Interval	Rocks Stopped
890 To 900 ft	0
900 To 910 ft	0
910 To 920 ft	0
920 To 930 ft	0
930 To 940 ft	0
940 To 950 ft	0
950 To 960 ft	0
960 To 970 ft	0
970 To 980 ft	0
980 To 990 ft	0
990 To 1000 ft	0
1000 To 1010 ft	0
1010 To 1020 ft	0
1020 To 1030 ft	0
1030 To 1040 ft	0
1040 To 1050 ft	0
1050 To 1060 ft	1
1060 To 1070 ft	0
1070 To 1080 ft	2
1080 To 1090 ft	2
1090 To 1100 ft	2
1100 To 1110 ft	0
1110 To 1120 ft	0
1120 To 1130 ft	0
1130 To 1140 ft	0
1140 To 1150 ft	0
1150 To 1160 ft	1
1160 To 1170 ft	0
1170 To 1180 ft	0
1180 To 1190 ft	0
1190 To 1200 ft	0
1200 To 1210 ft	0
1210 To 1220 ft	0
1220 To 1230 ft	0
1230 To 1240 ft	1
1240 To 1250 ft	0
1250 To 1260 ft	0
1260 To 1270 ft	0
1270 To 1280 ft	0
1280 To 1290 ft	0
1290 To 1300 ft	0
1300 To 1310 ft	0
1310 To 1320 ft	0
1320 To 1330 ft	0
1330 To 1340 ft	0
1340 To 1350 ft	0
1350 To 1360 ft	0
1360 To 1370 ft	0
1370 To 1380 ft	0

CRSP Input File - C:\Program Files\Crsp\D-D'.dat

Input File Specifications

Units of Measure: U.S.
Total Number of Cells: 12
Analysis Point X-Coordinate 1: 500
Analysis Point X-Coordinate 2:
Analysis Point X-Coordinate 3:
Initial Y-Top Starting Zone Coordinate: 8560
Initial Y-Base Starting Zone Coordinate: 8440
Remarks:

Cell Data

Cell No.	Surface R.	Tangent C.	Normal C.	Begin X	Begin Y	End X	End Y
1	5	.85	.35	0	8880	150	8800
2	5	.85	.35	150	8800	250	8720
3	5	.85	.35	250	8720	450	8560
4	1	.87	.37	450	8560	500	8440
5	5	.85	.35	500	8440	600	8320
6	5	.87	.35	600	8320	700	8240
7	5	.87	.35	700	8240	800	8160
8	5	.82	.33	800	8160	1050	8080
9	5	.82	.33	1050	8080	1250	8000
10	5	.82	.33	1250	8000	1450	7920
11	5	.82	.33	1450	7920	1600	7840
12	5	.82	.33	1600	7840	1850	7760

C:\Program Files\Crsp\D-D'.dat   Total Rocks Rolled: 25

Spherical Rock: 6-ft dia., 18661-lb

Scale: Each division = 20 feet

CRSP Analysis Point Data - C:\Program Files\Crsp\D-D'.dat

Analysis Point 1

Analysis Point 1: X = 500, Y = 8440

Spherical Rock: 6-ft dia., 18661-lb

Total Rocks Passing Analysis Point: 24

Velocity (ft/sec)

Maximum: 72.56
Average: 48.84
Minimum: 15.78
Std. Dev.: 15.07

Bounce Height (ft)

Maximum: 20.98
Average: 6.08
G. Mean: 3.63
Std. Dev.: 3.07

Kinetic Energy (ft-lb)

Maximum: 1664681
Average: 904902
Std. Dev.: 479381

CRSP Analysis Point Statistical Analysis - C:\Program Files\Crsp\D-D'.dat

Analysis Point 1

Analysis Point 1: X = 500, Y = 8440

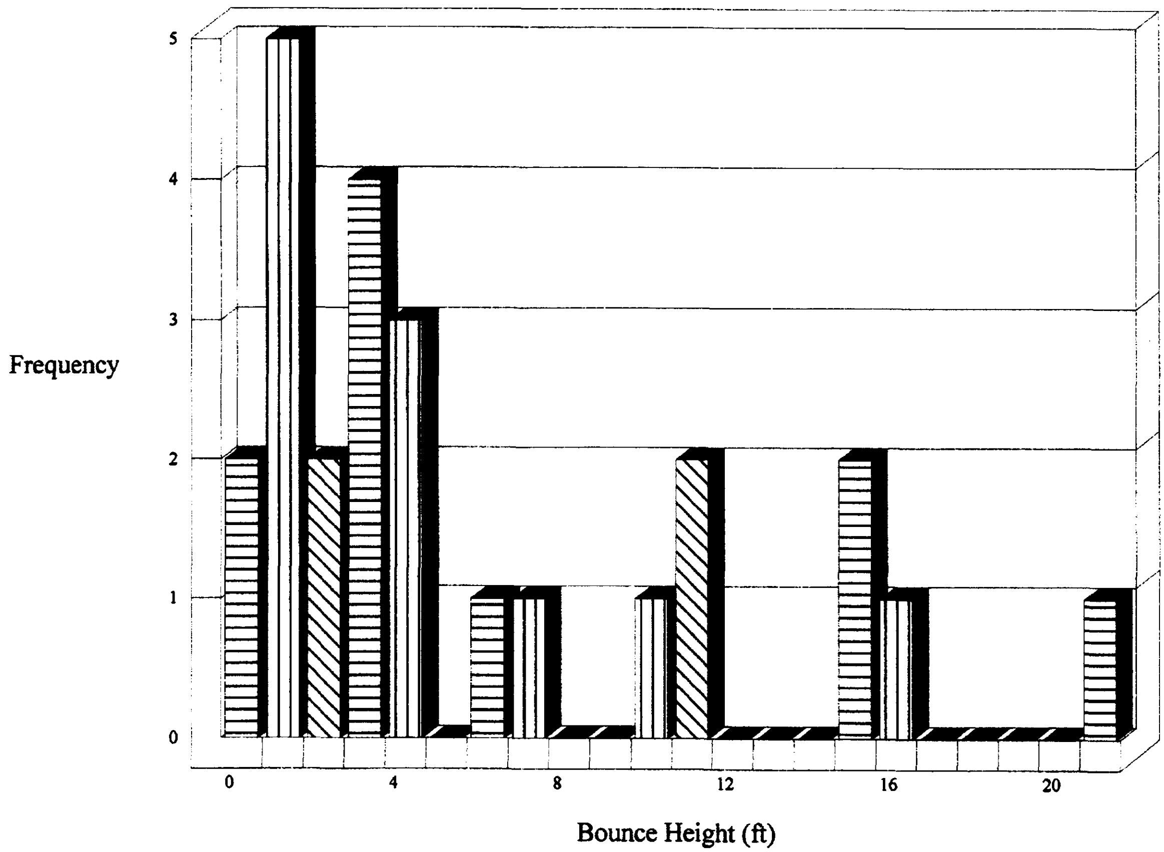
Spherical Rock: 6-ft dia., 18661-lb

Total Rocks Passing Analysis Point: 24

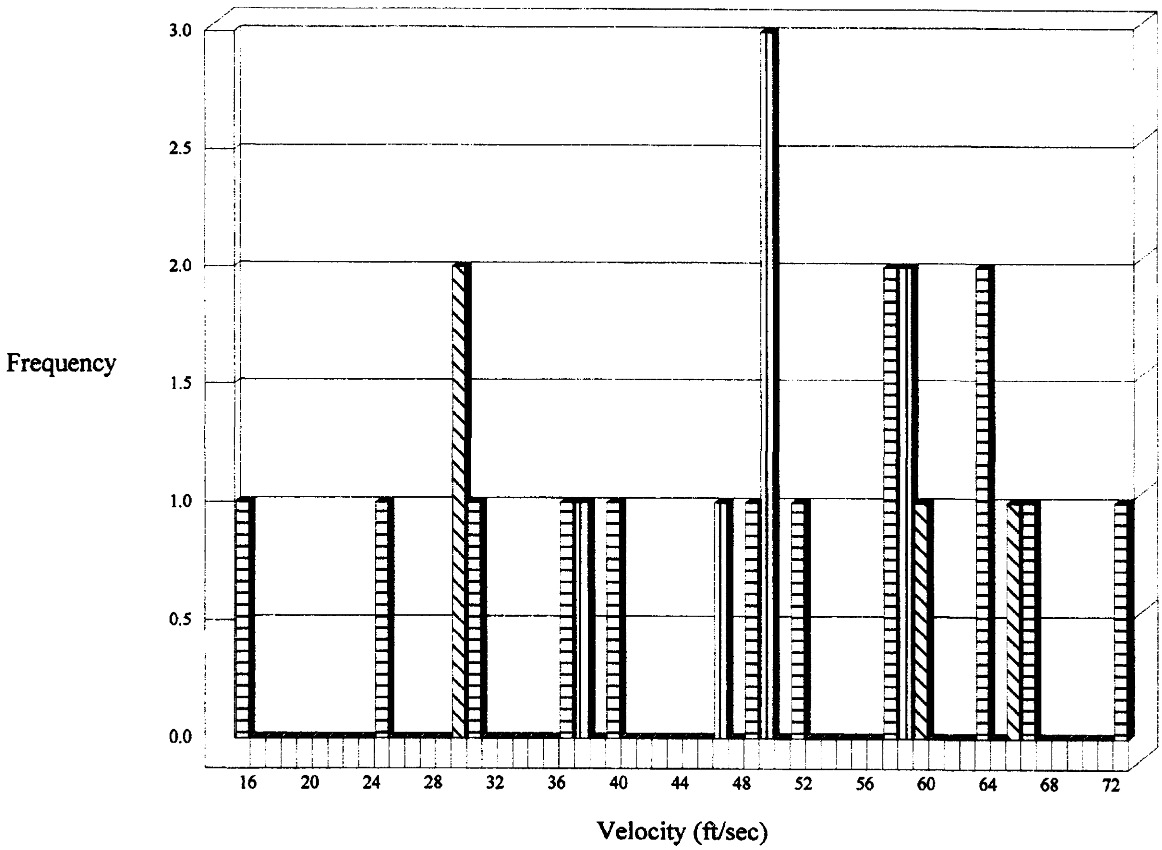
Cumulative Probability	Velocity (ft/sec)	Energy (ft-lb)	Bounce Height (ft)
50%	48.84	904902	3.63
75%	59.02	1228580	5.7
90%	68.17	1519708	7.56
95%	73.66	1694490	8.68
98%	79.83	1890653	9.94

Note: Velocity and kinetic energy are analyzed assuming a normal distribution.
Bounce height is analyzed assuming a log distribution.

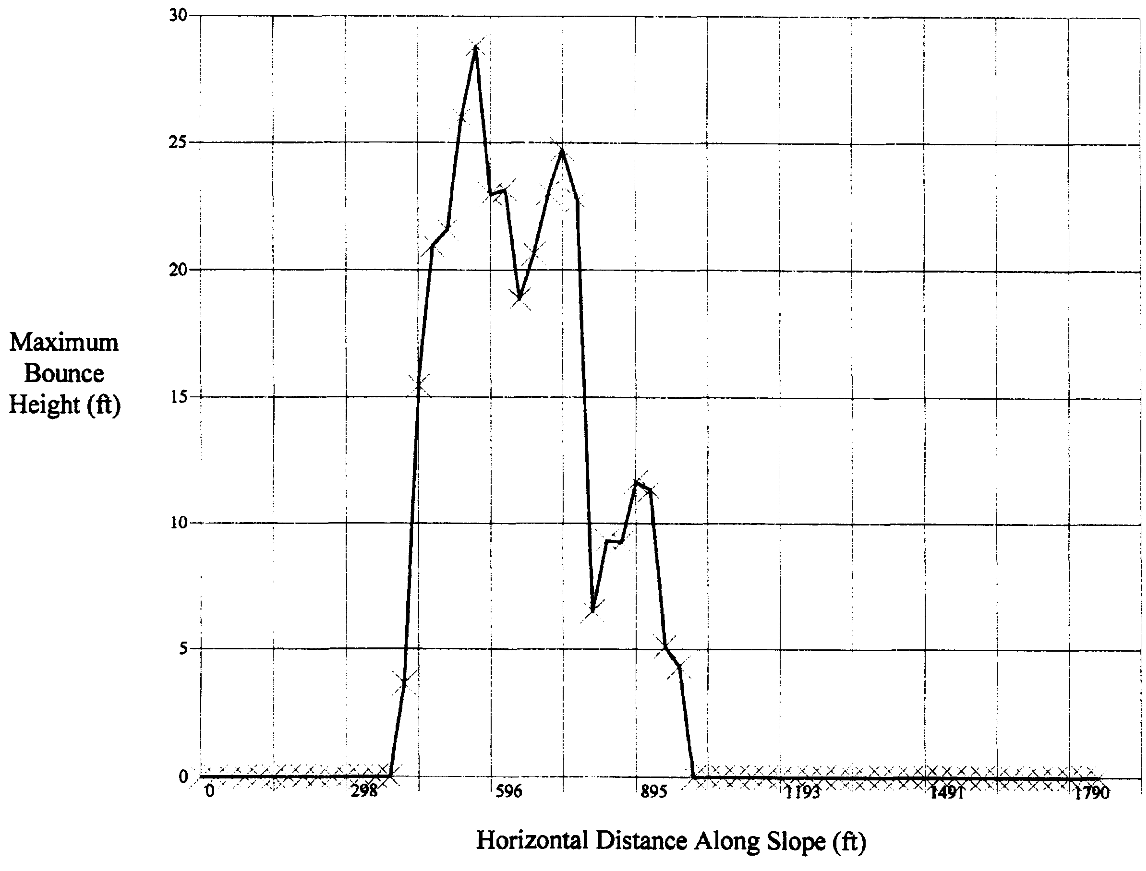
Analysis Point 1 Bounce Height Distribution - C:\Program Files\Crsp\D-D'.dat



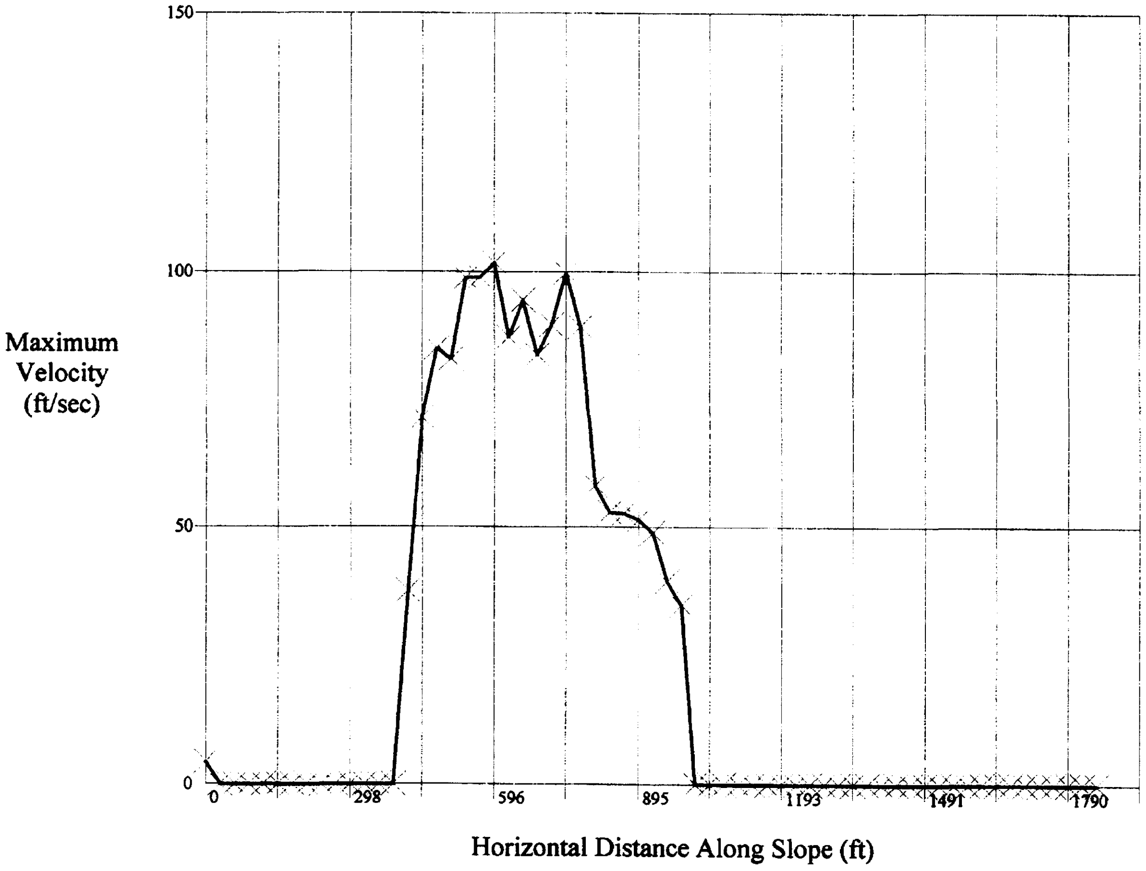
Analysis Point 1 Velocity Distribution - C:\Program Files\Crsp\D-D'.dat



Bounce Height Graph - C:\Program Files\Crsp\D-D'.dat



Velocity Graph - C:\Program Files\Crsp\D-D'.dat



CRSP Data Collected at End of Each Cell - C:\Program Files\Crsp\D-D'.dat

Velocity Units: ft/sec   Bounce Height Units: ft

Cell No.	Max. Velocity	Avg. Velocity	Std. Dev. Velocity	Max. Bounce Ht.	Avg. Bounce Ht.
1	No rocks	past end of cell			
2	No rocks	past end of cell			
3	No rocks	past end of cell			
4	73	49	15.07	21	6
5	91	59	16	29	11
6	91	50	20.45	17	6
7	79	46	17.61	23	7
8	No rocks	past end of cell			
9	No rocks	past end of cell			
10	No rocks	past end of cell			
11	No rocks	past end of cell			
12	No rocks	past end of cell			

CRSP Rocks Stopped Data - C:\Program Files\Crsp\D-D'.dat

X Interval	Rocks Stopped
0 To 10 ft	1
10 To 20 ft	0
20 To 30 ft	0
30 To 40 ft	0
40 To 50 ft	0
50 To 60 ft	0
60 To 70 ft	0
70 To 80 ft	0
80 To 90 ft	0
90 To 100 ft	0
100 To 110 ft	0
110 To 120 ft	0
120 To 130 ft	0
130 To 140 ft	0
140 To 150 ft	0
150 To 160 ft	0
160 To 170 ft	0
170 To 180 ft	0
180 To 190 ft	0
190 To 200 ft	0
200 To 210 ft	0
210 To 220 ft	0
220 To 230 ft	0
230 To 240 ft	0
240 To 250 ft	0
250 To 260 ft	0
260 To 270 ft	0
270 To 280 ft	0
280 To 290 ft	0
290 To 300 ft	0
300 To 310 ft	0
310 To 320 ft	0
320 To 330 ft	0
330 To 340 ft	0
340 To 350 ft	0
350 To 360 ft	0
360 To 370 ft	0
370 To 380 ft	0
380 To 390 ft	0
390 To 400 ft	0
400 To 410 ft	0
410 To 420 ft	0
420 To 430 ft	0
430 To 440 ft	0

X Interval	Rocks Stopped
440 To 450 ft	0
450 To 460 ft	0
460 To 470 ft	0
470 To 480 ft	0
480 To 490 ft	0
490 To 500 ft	0
500 To 510 ft	0
510 To 520 ft	0
520 To 530 ft	0
530 To 540 ft	0
540 To 550 ft	0
550 To 560 ft	0
560 To 570 ft	0
570 To 580 ft	0
580 To 590 ft	0
590 To 600 ft	0
600 To 610 ft	0
610 To 620 ft	0
620 To 630 ft	0
630 To 640 ft	0
640 To 650 ft	0
650 To 660 ft	0
660 To 670 ft	0
670 To 680 ft	0
680 To 690 ft	0
690 To 700 ft	0
700 To 710 ft	0
710 To 720 ft	0
720 To 730 ft	0
730 To 740 ft	0
740 To 750 ft	0
750 To 760 ft	0
760 To 770 ft	0
770 To 780 ft	0
780 To 790 ft	0
790 To 800 ft	0
800 To 810 ft	0
810 To 820 ft	3
820 To 830 ft	2
830 To 840 ft	1
840 To 850 ft	0
850 To 860 ft	0
860 To 870 ft	1
870 To 880 ft	2
880 To 890 ft	2

X Interval	Rocks Stopped
890 To 900 ft	0
900 To 910 ft	4
910 To 920 ft	2
920 To 930 ft	2
930 To 940 ft	0
940 To 950 ft	1
950 To 960 ft	0
960 To 970 ft	0
970 To 980 ft	0
980 To 990 ft	2
990 To 1000 ft	0
1000 To 1010 ft	0
1010 To 1020 ft	0
1020 To 1030 ft	1
1030 To 1040 ft	1
1040 To 1050 ft	0
1050 To 1060 ft	0
1060 To 1070 ft	0
1070 To 1080 ft	0
1080 To 1090 ft	0
1090 To 1100 ft	0
1100 To 1110 ft	0
1110 To 1120 ft	0
1120 To 1130 ft	0
1130 To 1140 ft	0
1140 To 1150 ft	0
1150 To 1160 ft	0
1160 To 1170 ft	0
1170 To 1180 ft	0
1180 To 1190 ft	0
1190 To 1200 ft	0
1200 To 1210 ft	0
1210 To 1220 ft	0
1220 To 1230 ft	0
1230 To 1240 ft	0
1240 To 1250 ft	0
1250 To 1260 ft	0
1260 To 1270 ft	0
1270 To 1280 ft	0
1280 To 1290 ft	0
1290 To 1300 ft	0
1300 To 1310 ft	0
1310 To 1320 ft	0
1320 To 1330 ft	0
1330 To 1340 ft	0
1340 To 1350 ft	0
1350 To 1360 ft	0
1360 To 1370 ft	0
1370 To 1380 ft	0

CRSP Input File - C:\Program Files\Crsp\E-E'.dat

Input File Specifications

Units of Measure: U.S.
Total Number of Cells: 8
Analysis Point X-Coordinate 1: 550
Analysis Point X-Coordinate 2:
Analysis Point X-Coordinate 3:
Initial Y-Top Starting Zone Coordinate: 8560
Initial Y-Base Starting Zone Coordinate: 8320
Remarks:

Cell Data

Cell No.	Surface R.	Tangent C.	Normal C.	Begin X	Begin Y	End X	End Y
1	5	.85	.35	0	8880	200	8800
2	5	.85	.35	200	8800	400	8640
3	5	.85	.35	400	8640	500	8560
4	1	.87	.37	500	8560	550	8320
5	5	.85	.35	550	8320	700	8240
6	5	.82	.33	700	8240	800	8160
7	5	.82	.33	800	8160	900	8080
8	5	.82	.33	900	8080	1000	8000

C:\Program Files\Crsp\E-E'.dat Total Rocks Rolled: 25

Spherical Rock: 6-ft dia., 18661-lb

Scale: Each division = 20 feet

CRSP Analysis Point Data - C:\Program Files\Crsp\E-E'.dat

Analysis Point 1

Analysis Point 1: X = 550, Y = 8320

Spherical Rock: 6-ft dia., 18661-lb

Total Rocks Passing Analysis Point: 24

Velocity (ft/sec)

Maximum: 112.98
Average: 64.91
Minimum: 18.89
Std. Dev.: 21

Bounce Height (ft)

Maximum: 68.68
Average: 19.43
G. Mean: 11.07
Std. Dev.: 3.51

Kinetic Energy (ft-lb)

Maximum: 3834550
Average: 1630396
Std. Dev.: 909983

CRSP Analysis Point Statistical Analysis - C:\Program Files\Crsp\E-E'.dat

Analysis Point 1

Analysis Point 1: X = 550, Y = 8320

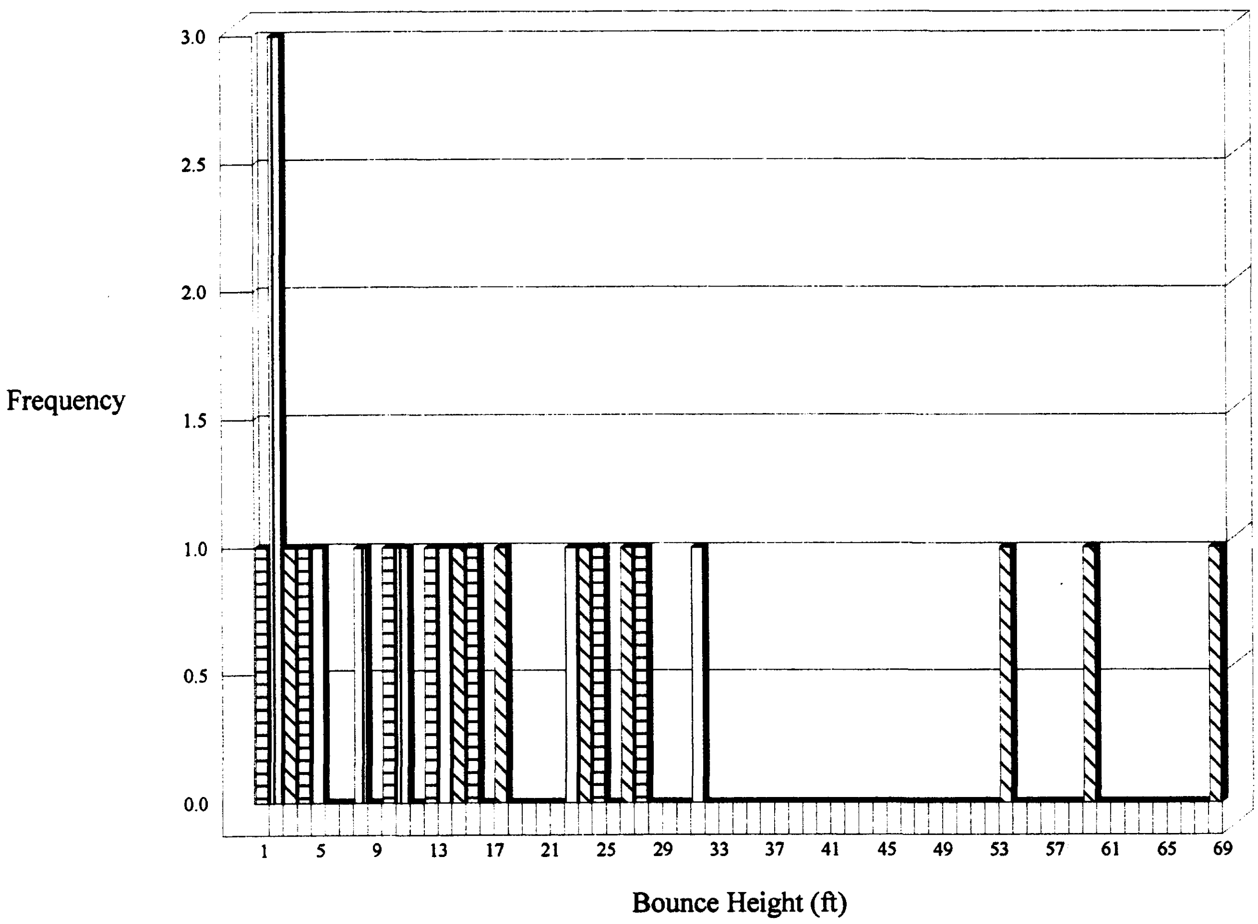
Spherical Rock: 6-ft dia., 18661-lb

Total Rocks Passing Analysis Point: 24

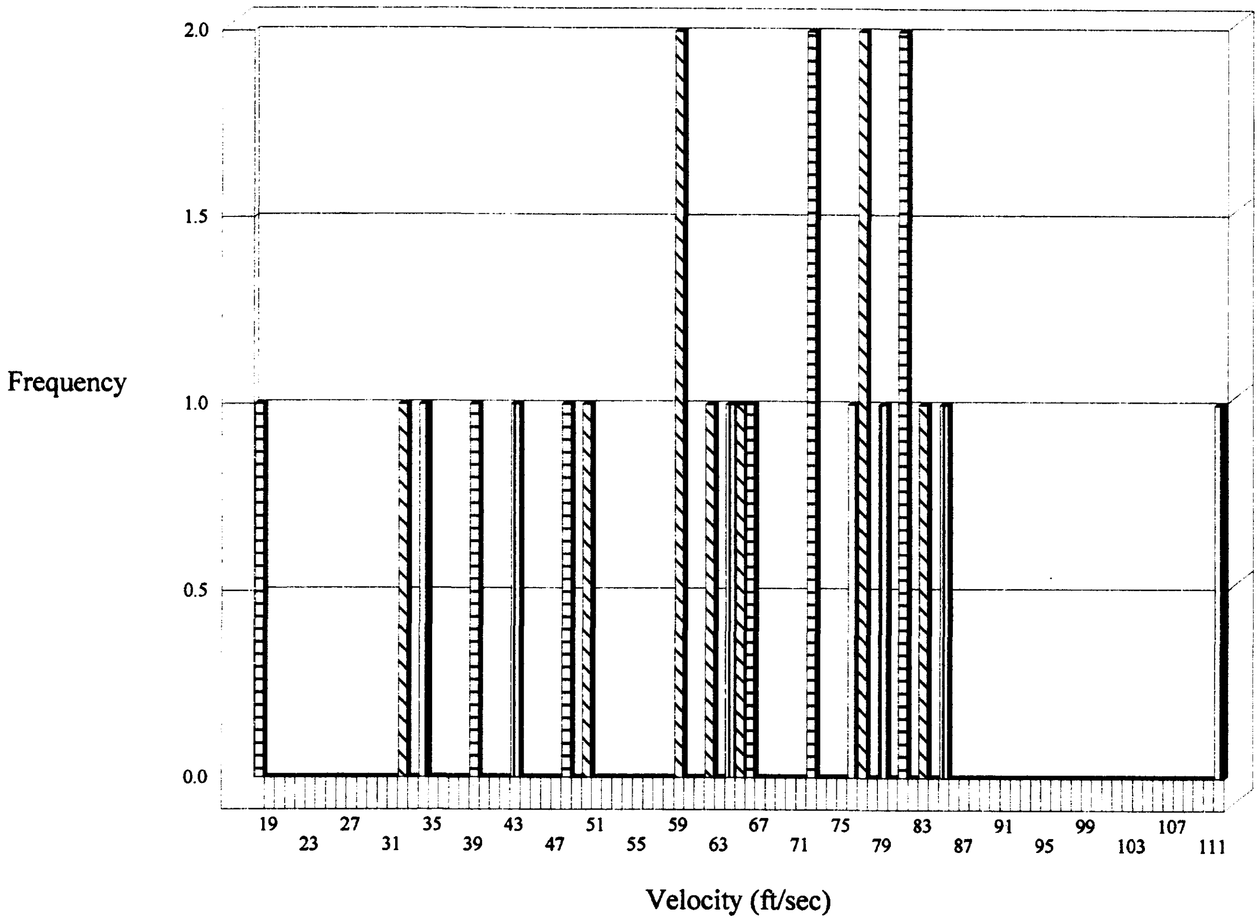
Cumulative Probability	Velocity (ft/sec)	Energy (ft-lb)	Bounce Height (ft)
50%	64.91	1630396	11.07
75%	79.09	2244817	13.44
90%	91.84	2797450	15.57
95%	99.5	3129230	16.85
98%	108.09	3501595	18.28

Note: Velocity and kinetic energy are analyzed assuming a normal distribution.
Bounce height is analyzed assuming a log distribution.

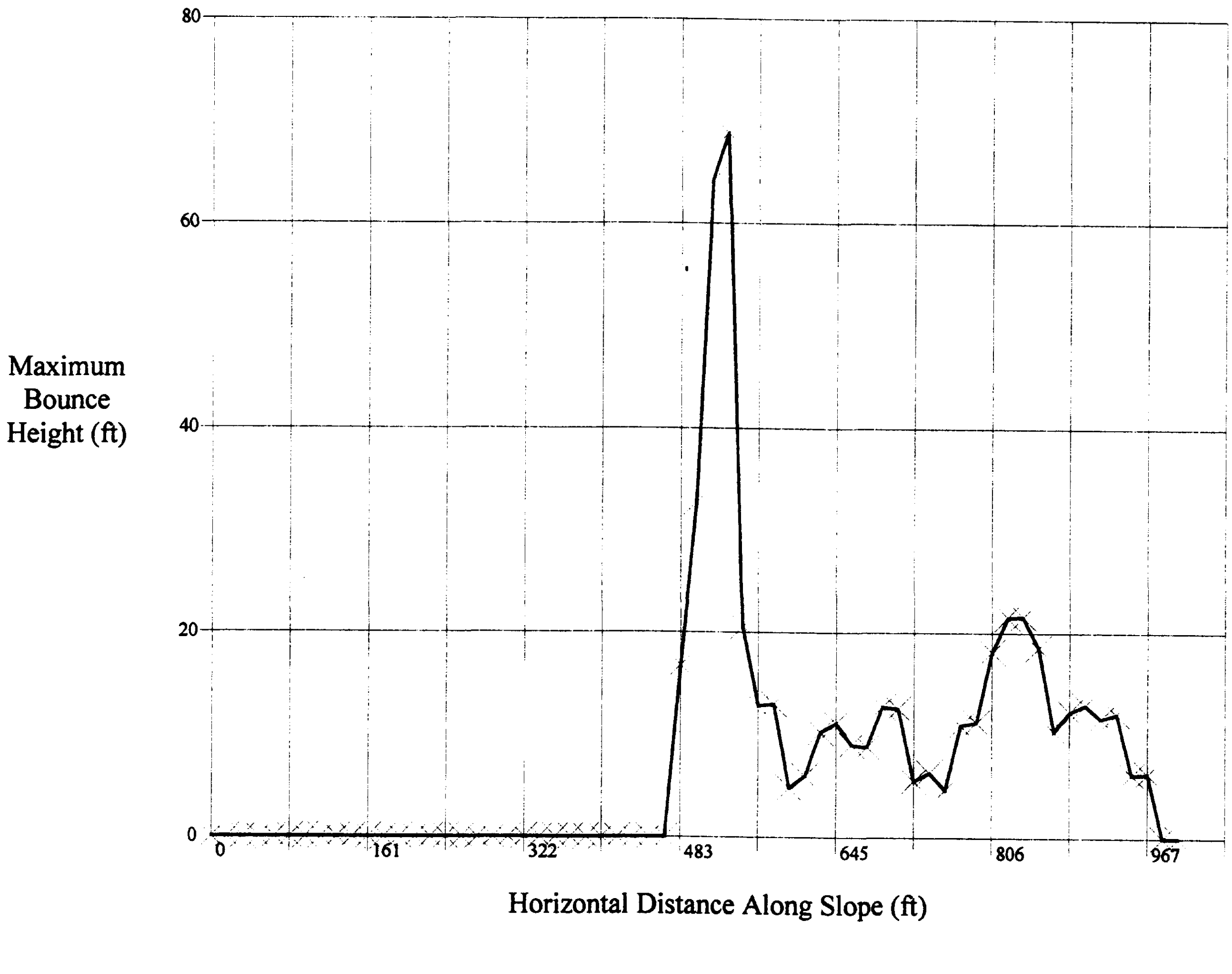
Analysis Point 1 Bounce Height Distribution - C:\Program Files\Crsp\E-E'.dat



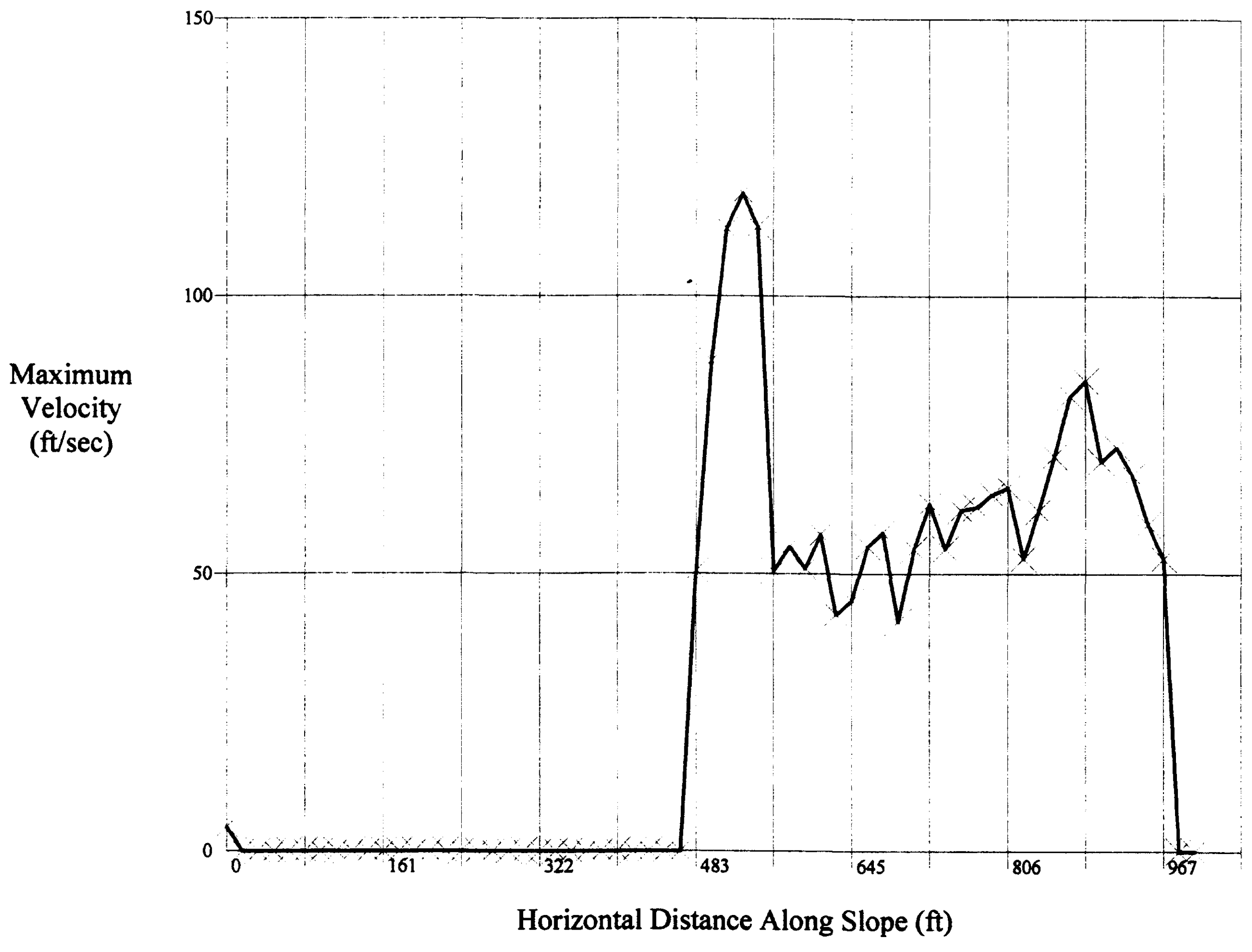
Analysis Point 1 Velocity Distribution - C:\Program Files\Crsp\E-E'.dat



Bounce Height Graph - C:\Program Files\Crsp\E-E'.dat



Velocity Graph - C:\Program Files\Crsp\E-E'.dat



CRSP Data Collected at End of Each Cell - C:\Program Files\Crsp\E-E'.dat

Velocity Units: ft/sec   Bounce Height Units: ft

Cell No.	Max. Velocity	Avg. Velocity	Std. Dev. Velocity	Max. Bounce Ht.	Avg. Bounce Ht.
1	No rocks	past end of cell			
2	No rocks	past end of cell			
3	No rocks	past end of cell			
4	113	65	21	69	19
5	55	28	17.13	4	2
6	55	36	11.39	6	3
7	72	51	14.99	12	6
8	52	33	12.88	5	4

CRSP Rocks Stopped Data - C:\Program Files\Crsp\E-E'.dat

X Interval	Rocks Stopped
0 To 10 ft	1
10 To 20 ft	0
20 To 30 ft	0
30 To 40 ft	0
40 To 50 ft	0
50 To 60 ft	0
60 To 70 ft	0
70 To 80 ft	0
80 To 90 ft	0
90 To 100 ft	0
100 To 110 ft	0
110 To 120 ft	0
120 To 130 ft	0
130 To 140 ft	0
140 To 150 ft	0
150 To 160 ft	0
160 To 170 ft	0
170 To 180 ft	0
180 To 190 ft	0
190 To 200 ft	0
200 To 210 ft	0
210 To 220 ft	0
220 To 230 ft	0
230 To 240 ft	0
240 To 250 ft	0
250 To 260 ft	0
260 To 270 ft	0
270 To 280 ft	0
280 To 290 ft	0
290 To 300 ft	0
300 To 310 ft	0
310 To 320 ft	0
320 To 330 ft	0
330 To 340 ft	0
340 To 350 ft	0
350 To 360 ft	0
360 To 370 ft	0
370 To 380 ft	0
380 To 390 ft	0
390 To 400 ft	0
400 To 410 ft	0
410 To 420 ft	0
420 To 430 ft	0
430 To 440 ft	0

X Interval	Rocks Stopped
440 To 450 ft	0
450 To 460 ft	0
460 To 470 ft	0
470 To 480 ft	0
480 To 490 ft	0
490 To 500 ft	0
500 To 510 ft	0
510 To 520 ft	0
520 To 530 ft	0
530 To 540 ft	0
540 To 550 ft	0
550 To 560 ft	5
560 To 570 ft	3
570 To 580 ft	3
580 To 590 ft	1
590 To 600 ft	0
600 To 610 ft	1
610 To 620 ft	0
620 To 630 ft	2
630 To 640 ft	2
640 To 650 ft	0
650 To 660 ft	0
660 To 670 ft	0
670 To 680 ft	1
680 To 690 ft	0
690 To 700 ft	1
700 To 710 ft	0
710 To 720 ft	0
720 To 730 ft	0
730 To 740 ft	0
740 To 750 ft	0
750 To 760 ft	0
760 To 770 ft	0
770 To 780 ft	0
780 To 790 ft	0
790 To 800 ft	0
800 To 810 ft	0
810 To 820 ft	0
820 To 830 ft	0
830 To 840 ft	0
840 To 850 ft	0
850 To 860 ft	0
860 To 870 ft	0
870 To 880 ft	0
880 To 890 ft	0

X Interval	Rocks Stopped
890 To 900 ft	0
900 To 910 ft	0
910 To 920 ft	0
920 To 930 ft	0
930 To 940 ft	0
940 To 950 ft	0
950 To 960 ft	0
960 To 970 ft	0
970 To 980 ft	0
980 To 990 ft	0
990 To 1000 ft	0

PREDICTION OF SURFACE DEFORMATION RESULTING FROM LONGWALL MINING OVER THE BEAR CANYON RESERVE

PREPARED FOR

C.W. MINING COMPANY

HUNTIGTON, UTAH

AUGUST 2006

PREPARED

BY

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1.0 INTRODUCTION

This report was prepared at the request of C.W. Mining Company for an evaluation of surface subsidence mechanics and determination of typical deformation expected at the Bear Canyon longwall reserve (figure 1) in the C.W. mining operations, located near Huntington, Utah. The study was initiated in response to a deficiency list prepared by resource specialists of the Utah Division of Oil, Gas, and Mining.

Specific objectives were as follows:

- Description of subsidence mechanisms and angle of draw;
- Description of pillar designs developed by C.W. Mining for the multiple seam reserve in the Bear Canyon Study area,
- Calculation of subsidence profiles over the longwall blocks in both Tank and Hiawatha seams using regional subsidence measurement results, and
- General recommendations for surface subsidence monitoring.

The study area is located adjacent to the permitted areas in the C.W. Mining existing room-and-pillar operations located in the Wasatch Plateau Coal Fields of eastern Utah. Longwall mining has been extensively used in both the Book Cliffs and Wasatch Plateau coal fields since its introduction at the Sunnyside mines during 1960's; it is generally considered an environmentally attractive method to mine coal. It minimizes damage to the surface by permitting gradual subsidence of overburden strata over mined-out areas while at the same time satisfying BLM requirements of maximizing economic recovery of coal resources (Maleki and others 2001).

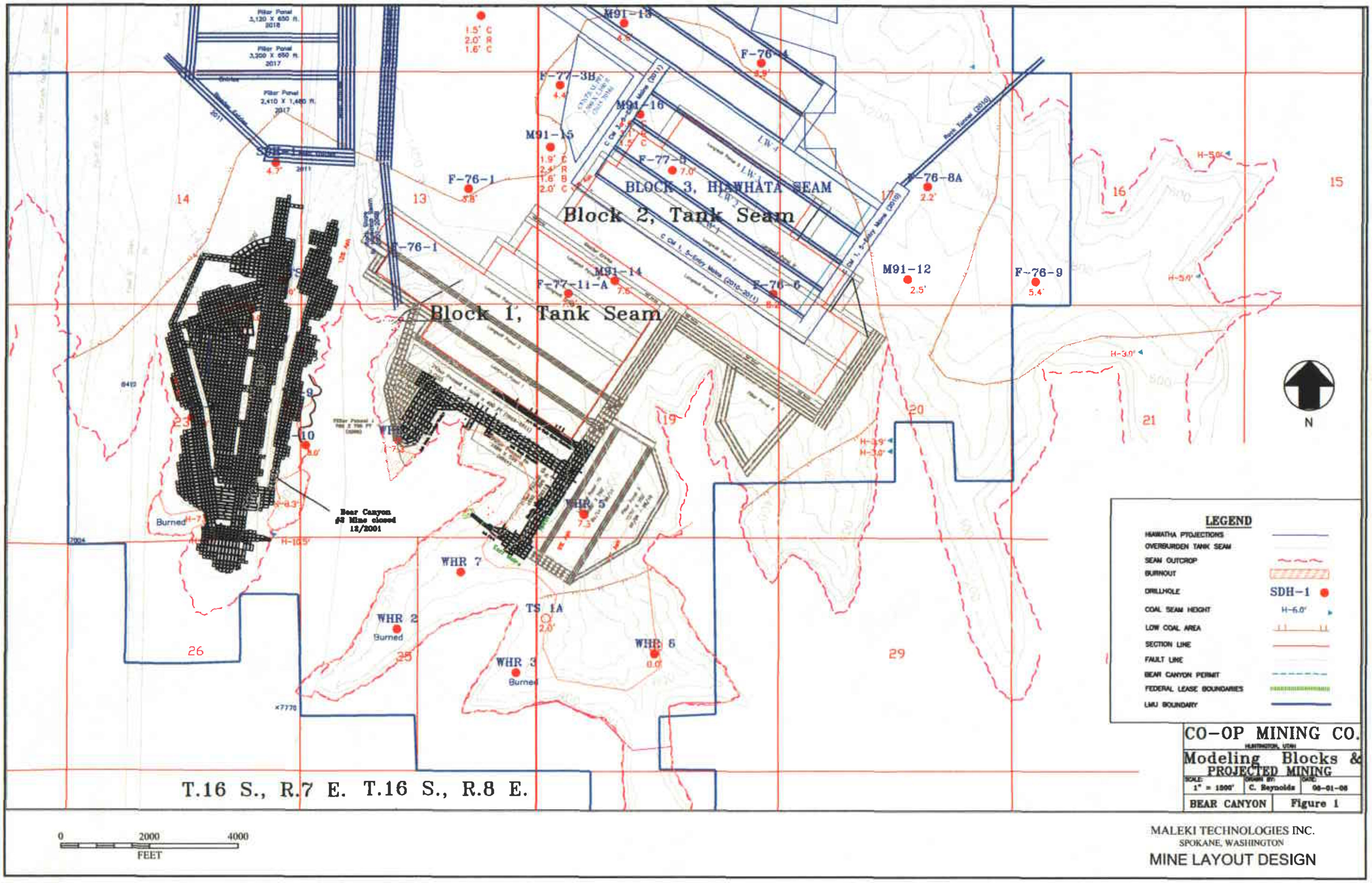
C.W. Mining is planning to mine coal reserves from the study area using the longwall method mostly in the Tank and Hiawatha seams at a typical depth of 800 to 2,000 ft (two limited panels are also envisioned in the Blind Canyon Seam). Existing mine plans call for extraction of the reserve using an extraction height of 5 to 8 ft within longwall panels. Subsidence calculations (consisting of vertical movements and horizontal strains) were completed for three longwall blocks, as illustrated in figure 1.

- Block 1, Tank Seam. For five 500- to-640-ft wide longwall panels retreated from northwest to southeast. Seam thickness varies from 5 to 7.6 ft within this longwall block. We have simulated an average extraction height of 7 ft. This is a conservative and prudent assumption for this study.

- Block 2, Tank Seam. For four 600- to-800-ft wide longwall panels retreated from northwest to southeast. Seam thickness varies from 5 to 8 ft within this longwall block. We have simulated an average extraction height of 7 ft.

- Block 3, Hiawatha Seam. For four 640-ft wide longwall panels retreated from northwest to southeast. Seam thickness varies from 5 to 15 ft within this longwall block. The extraction height is fixed at 8 ft considering longwall face equipment specifications.

This report is prepared in five sections. After this introduction, the subsidence mechanism is presented in section 2.0, followed in section 3.0 by a description of mining and geologic conditions and subsidence characteristics, including rock mechanics data, subsidence parameters, and a discussion of the conceptual mine layout designs developed by C.W. Mining for multiple seam longwall extraction. Predicted deformation patterns are presented in section 4 using three-dimensional subsidence models. The subsidence monitoring program is reviewed in section 5.



CO-OP MINING CO.
HUNTINGTON, UTAH

Modeling Blocks & PROJECTED MINING

SCALE:	DRAWN BY:	DATE:
1" = 1000'	C. Reynolds	06-01-06
BEAR CANYON	Figure 1	

MALEKI TECHNOLOGIES INC.
SPOKANE, WASHINGTON
MINE LAYOUT DESIGN

2.0 SUBSIDENCE MECHANISM

Surface subsidence occurs because of downward rock mass movement caused by the closure and collapse of mined-out excavations. Surface subsidence processes result in both vertical and horizontal displacement of rocks. Two major mechanisms of surface subsidence are associated with mining: formation of sinkholes and creation of troughs.

The type of subsidence mechanism predicted for the study area is the trough-type subsidence. It is characterized by the formation of a relatively smooth basin and is much less damaging than sinkhole subsidence. Sinkholes result from sudden or time-dependent collapse of overburden in localized areas, and these areas can be from several feet to tens of feet in diameter. Based on long-term measurements over the Hanna Basin, Wyoming (Karfakis 1987) and the Colorado Front Range (Matheson and Bliss 1986), researchers have established a relationship between the probabilities of sinkhole subsidence versus overburden depth. A great majority of sinkholes (98% probability) form where depths are less than 160 ft. At typical cover depths of 400 to 2,000 ft over the mains at the longwall project site, the probability of sinkhole occurrence is small, assuming stable "support" pillars.

As longwall operations are initiated in the first panel, roof span increases behind the longwall face until it caves. The roof span varies in mines, but typically ranges from 30 to 200 ft, depending on the strength of the roof rocks. The remaining overburden rocks will remain stable, transferring their load to the face and gate pillars. At some critical face position, the arching and load transfer mechanism collapses, and ground movement expands toward the surface, causing subsidence.

The caving process is associated with fracturing of near-seam strata and settling of overlying rocks. Four zones of movement are associated with subsidence (Peng 1992).

1. *Cave zone—broken and fragmented rocks that fill mined space.* The immediate roof rocks fracture into blocks often controlled by preexisting structure, filling the mined space. Bulking and rotation of individual roof rocks eventually limits the upward growth of failure. The thickness of this zone is estimated to be two to eight times seam thickness, depending on the bulking characteristics of the immediate roof rocks.

2. *Fracture zone—fractured rocks that fail because of shear stresses near the ribs and delamination toward the center of the panel.* This zone is located directly above the cave zone. The strata within this zone move downward, usually in large blocks, but without major rotation, to rest on the caved zone below. The permeability of the rocks is increased within this zone, which is estimated to extend twenty to sixty times seam thickness (Peng 1992) above the mine roof depending on geologic conditions and the strength of the rocks.

3. *Continuous deformation zone—deformation zone from the top of the fractured zone to the surface soils.* The strata flex downward without significant fracturing, gradually settling over the fracture zone. In the absence of soils, this zone extends to the surface,

forming compression zones at the surface to the center of the panel and tension zones at the edge of excavations.

4. *Soil zone*—This zone is an extension of the continuous deformation zone, which, depending on site-specific conditions, generally consists of soils and weathered rocks. Because of the less-brittle nature of soils, tensile cracks associated with transient subsidence may not be detected easily in front of the face and any existing fractures tend to heal quickly. Tensile fractures forming at panel boundaries last longer, but eventually get closed due to caving of fracture walls.

Three subsidence phases are associated with trough subsidence (figure 2).

1. The *subcritical phase* occurs immediately at the beginning when movement is in a small area at the center of the basin.

2. The *critical phase* occurs as the basin area expands when the maximum value of the downward movement is reached at the center. The critical excavation width is generally larger than 1.4 to 1.6 times the overburden thickness and is influenced by position and strength of competent layers within the overburden.

3. The *supercritical phase* occurs as the basin develops a flat bottom. In this phase, the basin area continues to increase with the cave area, but subsidence will remain at the maximum value attained in the critical phase.

Thus, the surface response of longwall mining activity, shown in Figure 2, begins with the subcritical phase, then progresses to the critical phase, and finally, to the supercritical phase. The subsidence process first shows effects on the surface as the upper strata bend, including tension (expansion), which causes near-surface fractures to open up and new ones to be created. Figure 2 shows how the middle portion of the excavation expands as subsidence continues, going through a cycle of, first, tension and then compression, which closes tension cracks. Final subsidence shows an excavation with the middle portions lower in elevation, but back to a near-original state. Areas on the edge of the excavation basin are subjected to tensile strains. Tensile strains are accumulative if the tensile zones overlap during the extraction of side-by-side panels ("transient" subsidence) or superimposed multiple-seam designs. By staggering the position of full extraction boundaries in multiple seams, C.W. Mining has avoided overlapping the tensile zones and thus reduced the potential for surface cracking at final mining boundaries ("permanent" subsidence).

Considering panel width to average overburden depth ratio for the C.W. Mining project area (0.6), these longwall panels are considered to have subcritical widths, and thus the great majority of subsidence is expected during the mining of the second and the third panels. The subsidence process is expected to be mature within 2 years after mining.

Subsidence characteristics for any coal field depends on site-specific geologic conditions and mining practices, including strata competence, geologic structure, topography, extraction height, extraction speed, and mine designs. For instance, rapid changes in topographic conditions are known to influence both naturally occurring and mining-induced rock mass wasting, including sandstone escarpment failure (Maleki and others 2001). The site-specific subsidence parameters for the Bear Canyon study area are addressed in the following sections using available monitoring results locally and regionally within Utah coal fields.

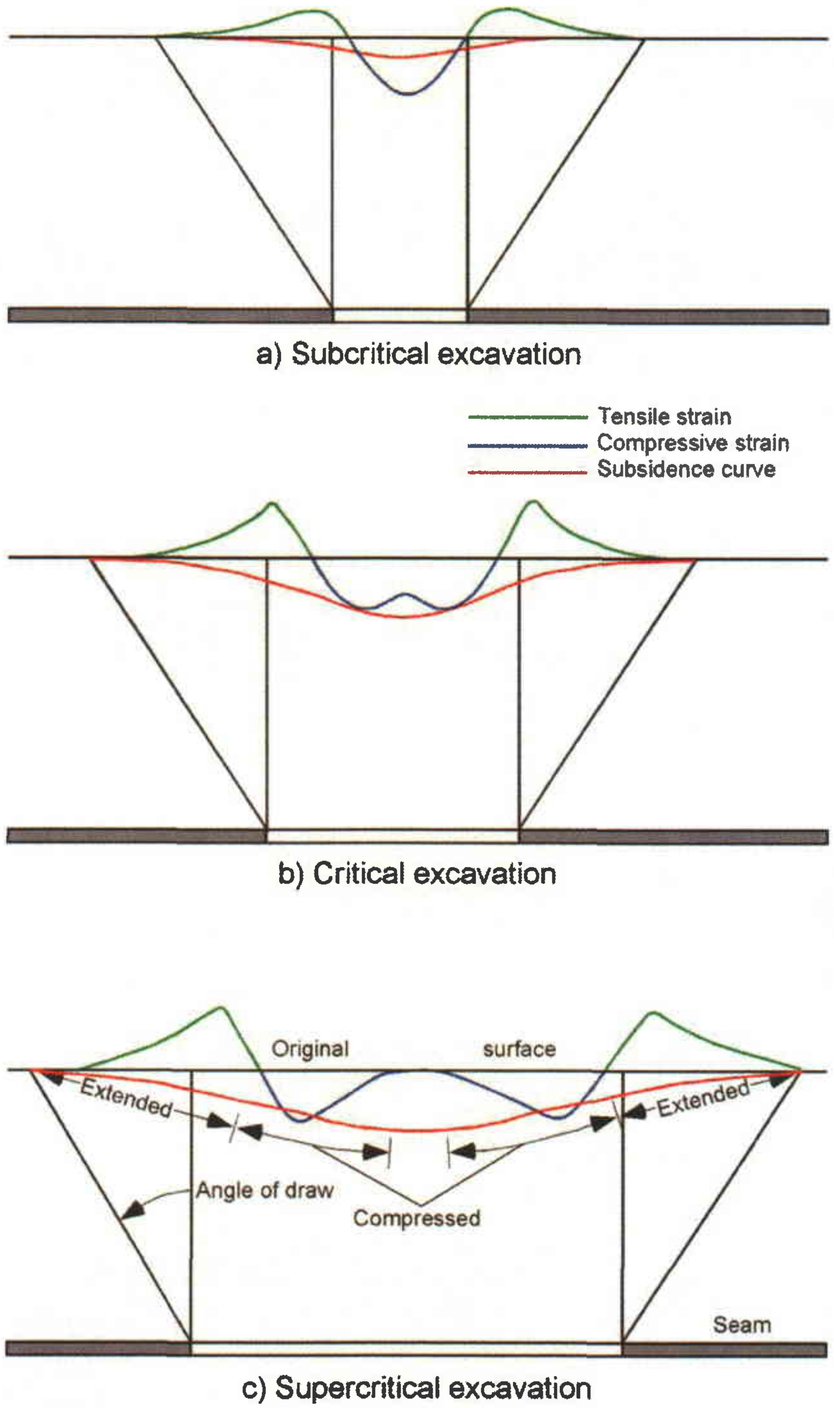


Figure 2. The three phases of subsidence development.

3.0 MINING, GEOLOGIC CONDITIONS AND SUBSIDENCE CHARACTERISTICS

3.1 Conceptual Mine Layout Designs

The C.W. Mining Company, in cooperation with MTI engineering staff, implemented geotechnical studies at both Tank and the Blind Canyon seams during the 1990's to study coal seam behavior and support loading during multiple seam pillar extraction (Maleki and others 1999). These studies consisted of underground and surface mapping, installation and monitoring of geotechnical instruments for the evaluation of Mobile Roof Support, and three-dimensional stress analyses. In addition, during 2001, MTI implemented a preliminary escarpment stability evaluation to assess potential pillaring impacts on the stability of the Castlegate Sandstone escarpments at the Wild Horse Ridge reserve. This study utilized a wealth of data collected over both stable and unstable escarpment areas at the neighboring East Mountain and Trail Mountain mines (Maleki and others 2000, MTI 2001).

As illustrated figure 1, C.W. Mining has oriented the longwall panels N55° W and is planning to use three-entry gateroad systems with 30-ft-wide yield pillars (50 ft center-to-center) and 500- to 800-ft-wide panels. This conceptual mine plan is suitable for permitting purposes and additional stress analyses are planned for finalizing mine designs in multiple-seam bump-prone conditions.

At sufficient deviation of 30° from major joint sets (N15° E and N85° E, MTI 2001), the existing mine orientation is beneficial for stability of development workings because it avoids alignment of joints and mine openings.

From environmental point-of-view, MTI considers this orientation effective in reducing the potential for subsidence-related cracking at the surface. Because at panel boundaries, the subsidence cracks generally form near parallel to longwall face and length (Maleki and others 2006), by misaligning the joints and panel orientation, C.W. Mining increases its chances of limiting the number and length of mining-induced surface fracturing at final mining boundaries.

To control gate pillar bumps, C.W. Mining staff has selected yield pillars to reduce strain energy accumulation within the gate pillars. Pillar size was selected on the basis of successful experience in the neighboring longwall operations in the East Mountain and Trail Mountain.

Based on a comprehensive case study by the USBM in 1980's (Fejes 1985, Dyni 1991, Section 4.2), MTI makes the assertion that the narrow 30-ft-wide yield pillars commonly used in the two-entry Utah reserves crush completely with no influence (or subsidence humps) above the gateroads. Thus the existing layout is also beneficial for reducing surface impacts, although we prefer a two-entry system for coal bump control based on site-specific geotechnical monitoring in the Dugout Canyon Mine (Maleki and others 2003).

3.2 Geology, Rock Strength and Stress Field

The three coal seams of economic interest belong to the Blackhawk Formation which is overlain by the Castlegate Sandstone and underlain by the Star Point Sandstone and the Mancos shale (figure 3). Minable longwall reserves are mostly contained in the Tank and Hiawatha seams with limited reserves also in the Blind Canyon Seam. Tank and Hiawatha seams average 7, and 8 ft in the study area; the Blind Canyon is 7 ft thick. The interburden between the Tank and Hiawatha seams is approximately 300-ft.

The overlying cliff forming Castlegate sandstone is a massive cross-bedded unit. It contains occasional thin, interbeds of shale, pebble conglomerate and mudstone. This unit is 170 to 250 ft thick in the area using the corehole data , however, the actual exposed thickness is locally much lower (as low as 50 ft). The Price River Formation consists of numerous beds of cross-bedded sandstones with occasional interbeds of shale, pebble conglomerate, and mudstone.

The Blackhawk Formation is composed of interbedded deltaic mudstone and siltstone and is less resistant to weathering than the neighboring units. It is characterized by alternating slope and cliff forming units. This unit is approximately 750 ft in thickness.

The Star Point Sandstone consists of thick cliff-forming sandstone units separated by shales. It is light colored and is approximately 350 ft in thickness in the study area. The Mancos shale is a blue-grey color marine shale, approximately 1000 ft thick, and is soft and well weathered.

Jointing patterns were mapped at the Castlegate Sandstone horizon and found similar across the study area (MTI 2001). The joint trends are thought to be generally coincident with jointing found in the overlying Price River and underlying Blackhawk Formations and are consistent with the measurements on the Wasatch Plateau (Maleki 1988, Maleki and others 1999). Joints were typically within a few degrees from vertical.

The most pronounced (primary) joint trend typically ranges between N10° E to N20° E (N15° E average). A less pronounced and secondary joint system trending S80° E to S90° E was also observed. This trend appeared to be generally consistent across the study area.

A third joint set was observed infrequently with a N50° E to N55° E trend. This set was only observed in the east near the Fish Creek Canyon. Spacing on this set is estimated to be greater than 10 feet due to its lack of occurrence or expression.

Apparent joint spacing appears to be controlled by confining stress. In outcrop the primary and secondary joints are more apparent and appear closer spaced at or near the points than in head of drainages. Rocks in place often exhibit jointing at 10 - 15 feet spacing, but more broken rocks nearly always showed closer spaced joints.

Faulting is not expected within the longwall reserves (Reynolds 2006).

Site-specific geologic and rock mechanics data are limited, although MTI has collected large amounts of information from adjacent properties. Figure 4 summarize the mechanical properties of coal measure strata at the neighboring East Mountain, compiled by MTI. Clearly, most overburden rocks are strong and stiff, capable of accumulating large strain energies, which contributes to seismicity.

The researchers from the former USBM and private industry have made a number of stress measurements in mines of Wasatch Plateau, Utah. There are two stress measurements within the close proximity of C.W. Mining operations (figure 5). These measurements confirm that the far-field stress field is unremarkable. The horizontal stress is moderate and is less than 50 percent of the vertical stress magnitude. We anticipate similar stress field at the C.W. Mining operations based on observations of lack of stress-induced stability problems (such as cutters) and an analyses of measurements in the existing reserve (Maleki and others 2000).

3.3 Subsidence Parameters

Subsidence engineering parameters include subsidence factor, angle of draw, angle of critical deformation, and horizontal strain. The subsidence factor is the ratio of maximum measured subsidence to extraction height. Because this ratio depends on excavation width and overburden thickness, it should be measured in supercritical excavations where caving has reached the surface on collapse of the pressure arch.

The angle of draw defines the limit of surface movements beyond the edge of an excavation. It is measured from a vertical line drawn at the panel edge and a line connecting the panel edge to the point of "no" movement on the surface. In practice, the accuracy of surveying equipment defines the point of no movement. This accuracy is usually about 0.1 ft but varies depending on topographic conditions, measurement technique, etc. Angle of critical deformation is similar to the angle of draw, but is measured to a point of critical deformation with respect to existing structures; it is preferred by many practitioners because it avoids the shortfalls connected with the accuracy of surveying equipment. Based on subsidence data from 40 longwall panels, Peng (1992) found that it is 10° less than the angle of draw.

Horizontal strain is the change in horizontal length of the ground divided by the original length of the ground. Positive strain is used here to show tensile strain indicating an increase in the horizontal length of the ground. Compressive strain (negative notation) occurs when the ground is shortened or compressed. Maximum tensile strain is found in supercritical excavation and maximum compressive strain occurs in subcritical excavations. Horizontal strain increases with an increase in extraction height and decreases at greater depths. Surface topography also influences horizontal strain.

The best estimates for the extend and magnitude of subsidence for the C.W. Mining two-seam mining conditions can be obtained by reviewing the results of long-term monitoring in Utah. The USBM implemented a comprehensive subsidence study over the Energy West two-seam longwall reserve from 1978 to 1989. The study monitored surface movements over four Blind Canyon and six Hiawatha panels. The study addressed angle of draw, subsidence factors for single and multiple-seam mining, and critical width. Similar to the Bear Canyon reserve, the mining area was bounded by faults. Maximum subsidence was 68% to 72% of the extraction height for single and two-seam mining conditions, respectively. This is in general agreement with other measurements in Utah showing a subsidence factor of 70 %. The angle of draw ranged between 25° to 30° for single- and two-seam mining conditions, respectively. This reported maximum angle of draw is higher than average values for the East Mountain (22.5° to 25°, Fejes 1985) but is significantly lower than that reported by the British National Coal Board (NCB 1975).

3.4 Gate Pillar Behavior

Because gate pillar designs may influence surface subsidence, some recent investigations have focused on evaluating subsidence above gate pillars. The Western U.S. measurements show different overburden deformation characteristics influenced by the choice of pillar designs. Based on a comprehensive case study by the USBM in 1980's (Fejes 1985, Dyni 1991, Section 4.2), MTI makes the assertion that the narrow 30-ft-wide yield pillars commonly used in the two-entry Utah reserves crush completely with no influence (or subsidence humps) above the gateroads.

We expect the three-entry yield pillar system at the Bear Canyon Mine to behave similarly in the long-term with no subsidence humps. The exact timing of pillar crushing is uncertain at this time requiring additional stress analyses. However, based on geotechnical measurements in Utah coal fields (Maleki and others 2003), pillar yielding to residual strength can occur quietly rapidly behind the face in moderately deep mines. We expect pillar crushing to be complete after extraction of panels on both sides because of significant convergence at the seam horizon. Site-specific calculations to address ground control issues in the three-entry system are forthcoming and will form the basis for petition to switch to a two-entry system.

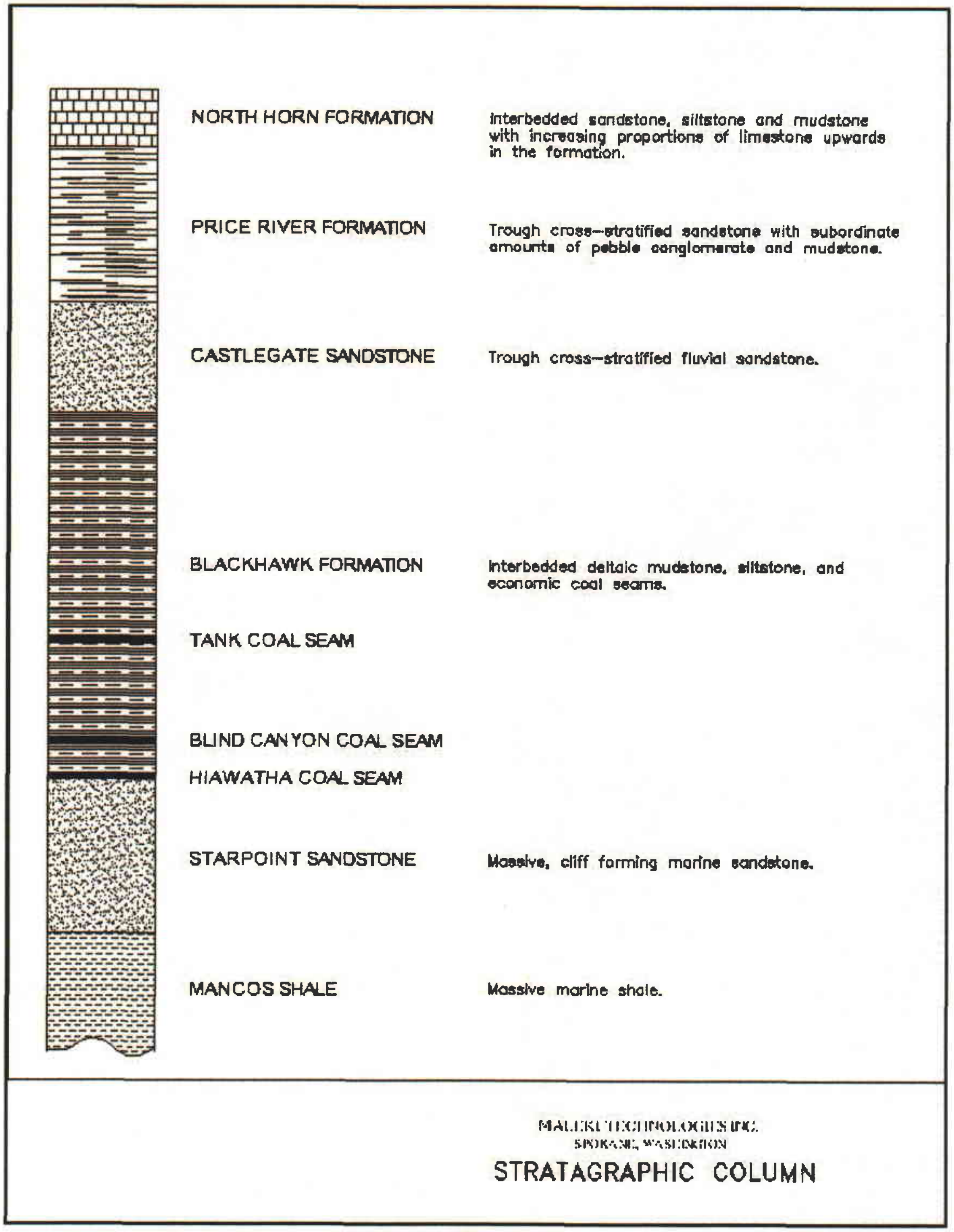
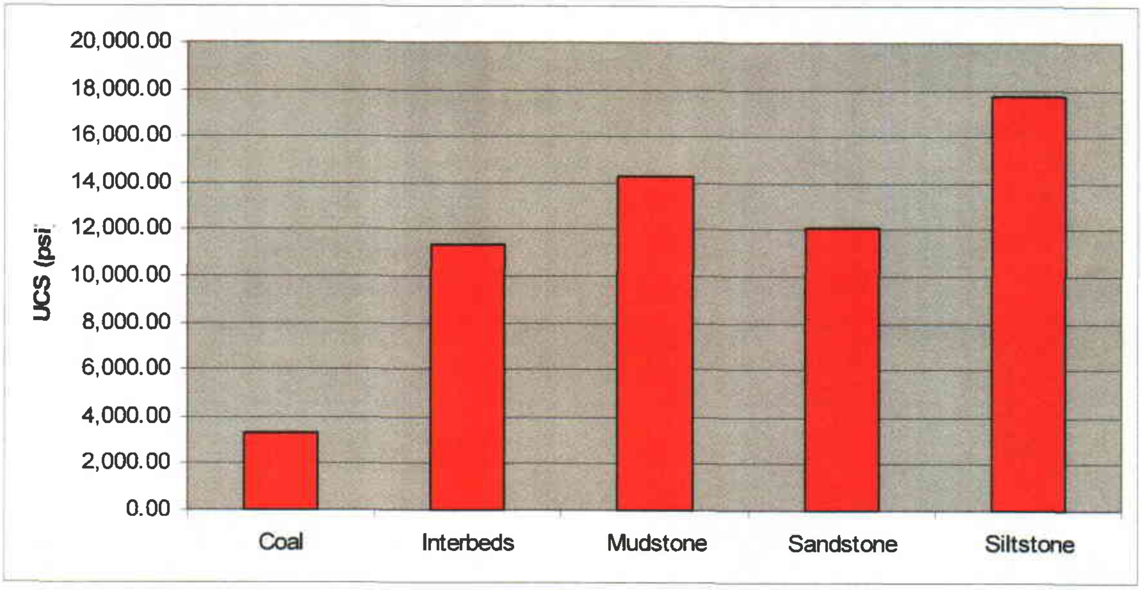


Figure 3. Generalized stratigraphic column.



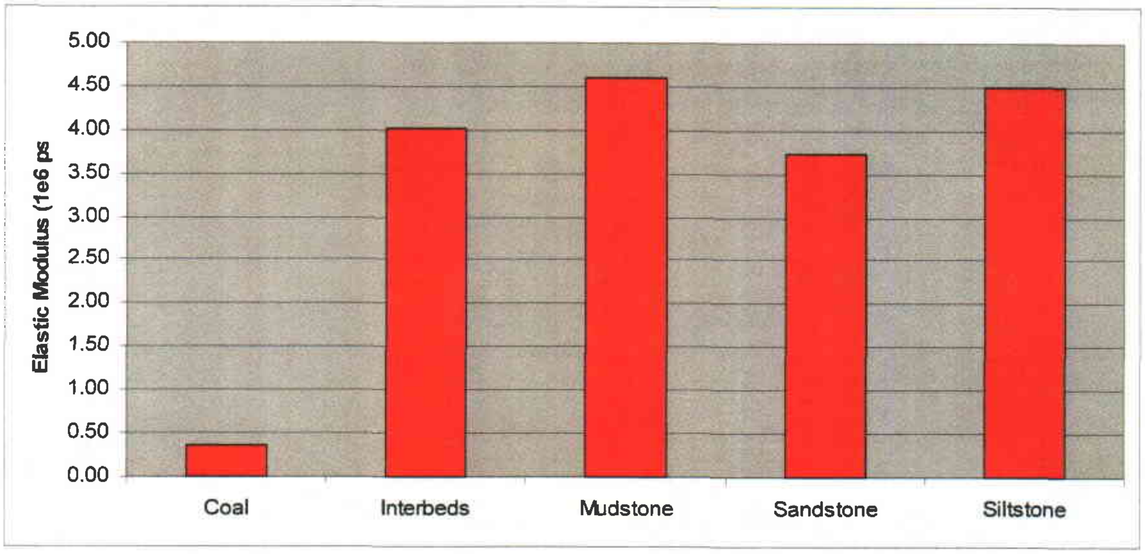


Figure 4. Histogram frequency diagram of uniaxial compressive strength and Young's modulus, regional data compiled by MTI.

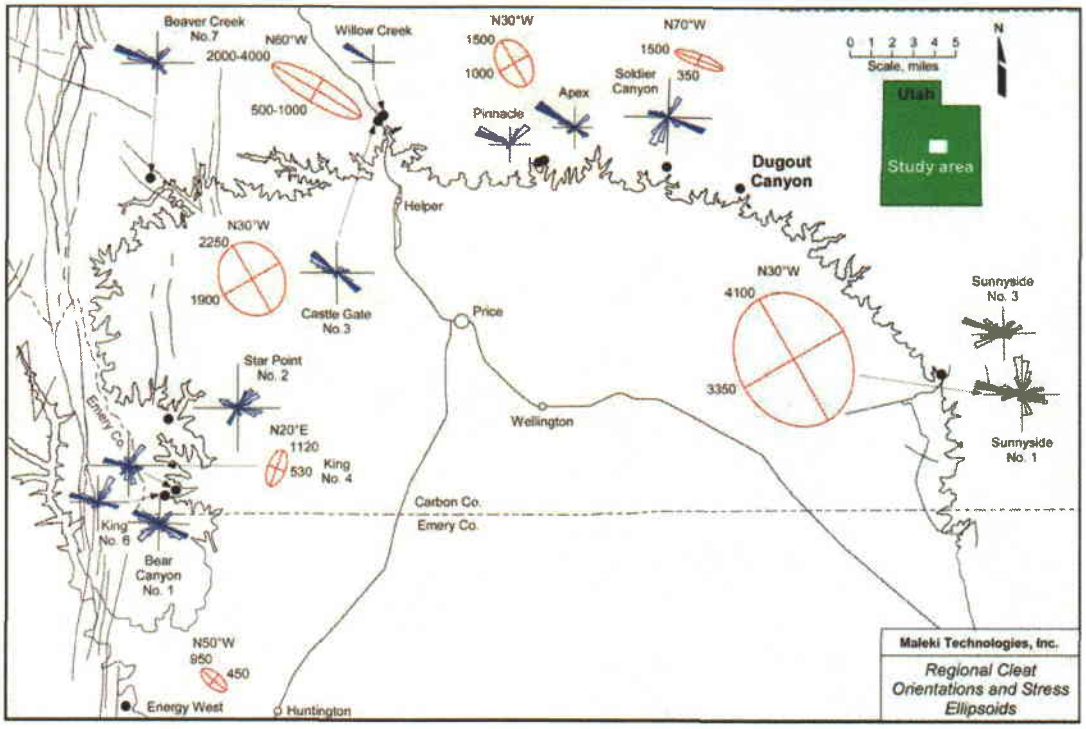


Figure 5. Regional horizontal stress measurements (ellipsoids) and the orientation of cleats in Utah coal fields.

4.0 PREDICTED GROUND MOVEMENTS

4.1 Methodology

Surface subsidence is the readily observable manifestation on the ground surface of the displacement field surrounding the underground portion of the mine. Predicting subsidence magnitude, therefore, constitutes a particular solution of the overall problem of finding the induced displacement field. To study subsidence phenomena and estimate the magnitude of subsidence, a number of empirical, physical, and numerical methods have been used.

Empirical methods, including profile functions, influence functions, and graphical methods were proposed by the British National Coal Board. These methods involve the analysis of existing subsidence from an area to predict future subsidence effects. These methods are based on the mathematical fit of a considerable number of measured subsidence profiles. They apply to geologic conditions in the area where they were developed and require adjustments if they are applied to different strata conditions.

To estimate surface deformation above the proposed longwall panels, we used a three-dimensional influence function method while accounting for site-specific conditions using the subsidence monitoring data from both the neighboring Deer Creek Mine. These methods have become very popular for the prediction of subsidence and surface strains within the last two decades (USBM, 1983; Peng and others 1994; SDPS 2000). They are superior to graphical methods because they can be used to model an entire longwall block while allowing an examination of the sensitivity of results to variations in seam thickness, pillar designs, panel dimensions, and overburden thickness.

These methods rely on the influence of an extracted volume on the displacement components of a remote point on the surface. In the zone calculation method, for example, the circular zone of influence around a point on the ground surface is divided into a number of zones in such a manner that the influence factor of such an area is fixed at a certain value. If the full area of the influence were mined out, the point in question would undergo 100% of maximum possible subsidence. If some portion within the zone of influence were unmined, subsidence would be correspondingly reduced.

Subsidence calculations (consisting of vertical movements, change in surface slopes and strains) were completed for three longwall blocks.

- Block 1, Tank Seam. For five 500- to-640-ft wide longwall panels retreated from northwest to southeast. Seam thickness varies from 5 to 7.6 ft within this longwall block. We have simulated an average extraction height of 7 ft. This is a conservative and prudent assumption for this study.

- Block 2, Tank Seam. For four 600- to-800-ft wide longwall panels retreated from northwest to southeast. Seam thickness varies from 5 to 8 ft within this longwall block. We have simulated an average extraction height of 7 ft.

- Block 3, Hiawatha Seam. For four 640-ft wide longwall panels retreated from northwest to southeast. Seam thickness varies from 5 to 15 ft within this longwall block. The extraction height is fixed at 8 ft considering longwall face equipment specifications (Reynolds 2006).

4.2 Model Calibration

Subsidence predictions were made using a numerical model calibrated with baseline subsidence data from the East Mountain. The long-term surface response to longwall mining in 5E, 6E, 7E and 8E panels was monitored by researchers from the U.S. Bureau of Mines in two phases (figure 6). These panels were mined from May 1974 through January 1983, and subsidence was monitored along five monument lines from September 1979 to June 1983 during phase 1 investigations. Phase 2 results reported by Dyni (1991) include surface response to mining the Hiawatha Seam some 60-ft below extracted 5E through 8E panels in the Blind Canyon Seam.

USBM study reports an average angle of draw of 25 degrees ranging from 16 to 33 degrees, and a final subsidence factor of 67 percent for single-seam mining. Surface effects were described as follows (Fejes 1985):

"*There were no visual surface effects within the subsidence area. The local vegetation were not altered, and no surface fissures were detected……*"

The results of phase 1 monitoring were used to establish modeling parameters. Figures 7 and 8 present a comparison of measured and calculated subsidence along a north-south monument line during the extraction of each four longwall panels and show good agreement. The subsidence factor increased from .35 during the extraction of 6E to 0.67 after the extraction of 8E. Note that yielding gate pillars used in this longwall block, crushed uniformly, showed no humps in the subsidence trough.

The calibrated version of the model was used to make quantitative predictions of the subsidence expected over the Bear Canyon Mine. The similarities in geology and geometry (depth of cover, face width, yielding gate pillars, and mining height) between the monitored area over East Mountain and the neighboring project area justify the use of the back-analyzed parameters for the predictive model.

Some uncertainty exists for predictions made with the model due to variations in geology and mining geometry, including actual mining heights. Precise estimates of subsidence can only be achieved as site-specific data become available, and mine plans are finalized.

4.3 Results

Figure 9 presents expected subsidence pattern after the extraction of each longwall block and figure 10 the combined two-seam subsidence resulting from extraction of blocks 1, 2, and 3 after the completion of mining in the Tank and Hiawatha seams.

Figures 11 and 12 present both subsidence and surface strain profiles along section A passing through the two-seam longwall extraction zone. Additional results are summarized in table 1 including changes in surface slopes.

Table 1. Predicted subsidence parameters for single and two-seam extraction design options.					
Block	Average mining height, ft	cover, ft	Maximum subsidence, ft	Maximum tensile strain, ft/ft	Maximum slope, percent
1 Tank	7	1,000	4.9	3.2e-3	.7
2 Tank	7	1,000	4.9	3.2e-3	.7
All combined	8	1,300	10.4	3.2e-3	1

Predicted subsidence varies from approximately4.9 to 10.4 ft for single and two-seam extraction. Using a criterion suggested by Singh and Bhattacharya (1984), tensile strains do not reach levels that could cause localized surface fracturing except at shallow areas (<650-ft cover). This assertion is in agreement with USBM measurements and observations on the East Mountain. The potential for surface fracturing is not higher at the permanent two-seam boundaries because longwall layouts in the Tank and Hiawatha seams are staggered. By not columnizing the longwall extraction areas in multiple seams and by not aligning panel orientation with primary joints, C.W. Mining has reduced the potential for surface fracturing.

Expected surface movement beyond underground mining boundaries varies from 460 ft in block 1 to 750 ft to the northeast where two-seam mining is planned in blocks 2 and 3. Changes in surface slopes are small (approximately one percent).

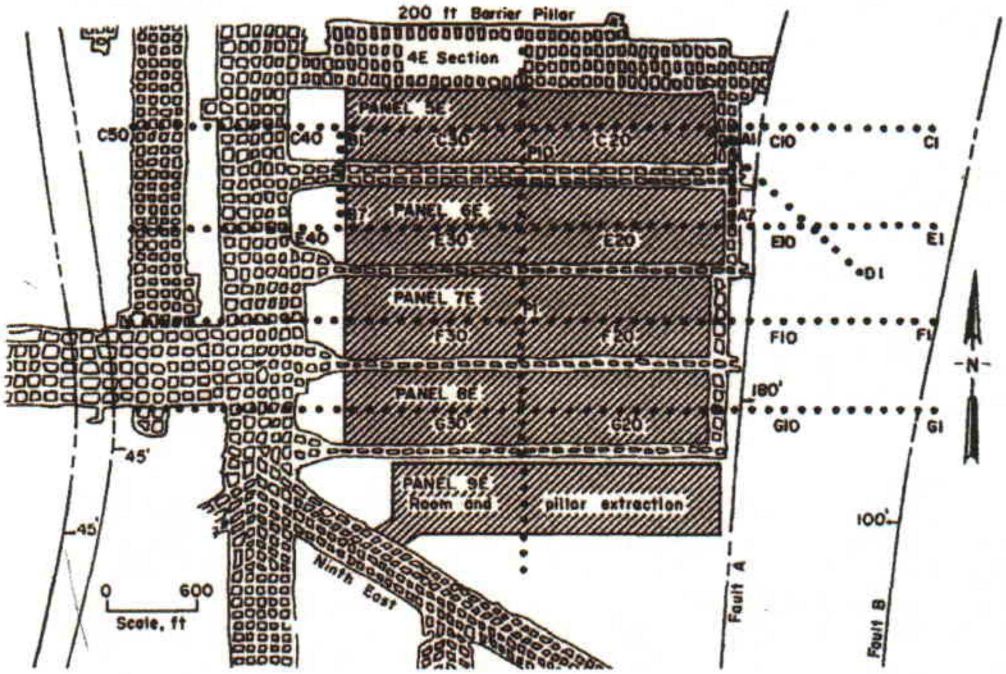


Figure 6. Subsidence monument locations above the USBM study site, East Mountain.

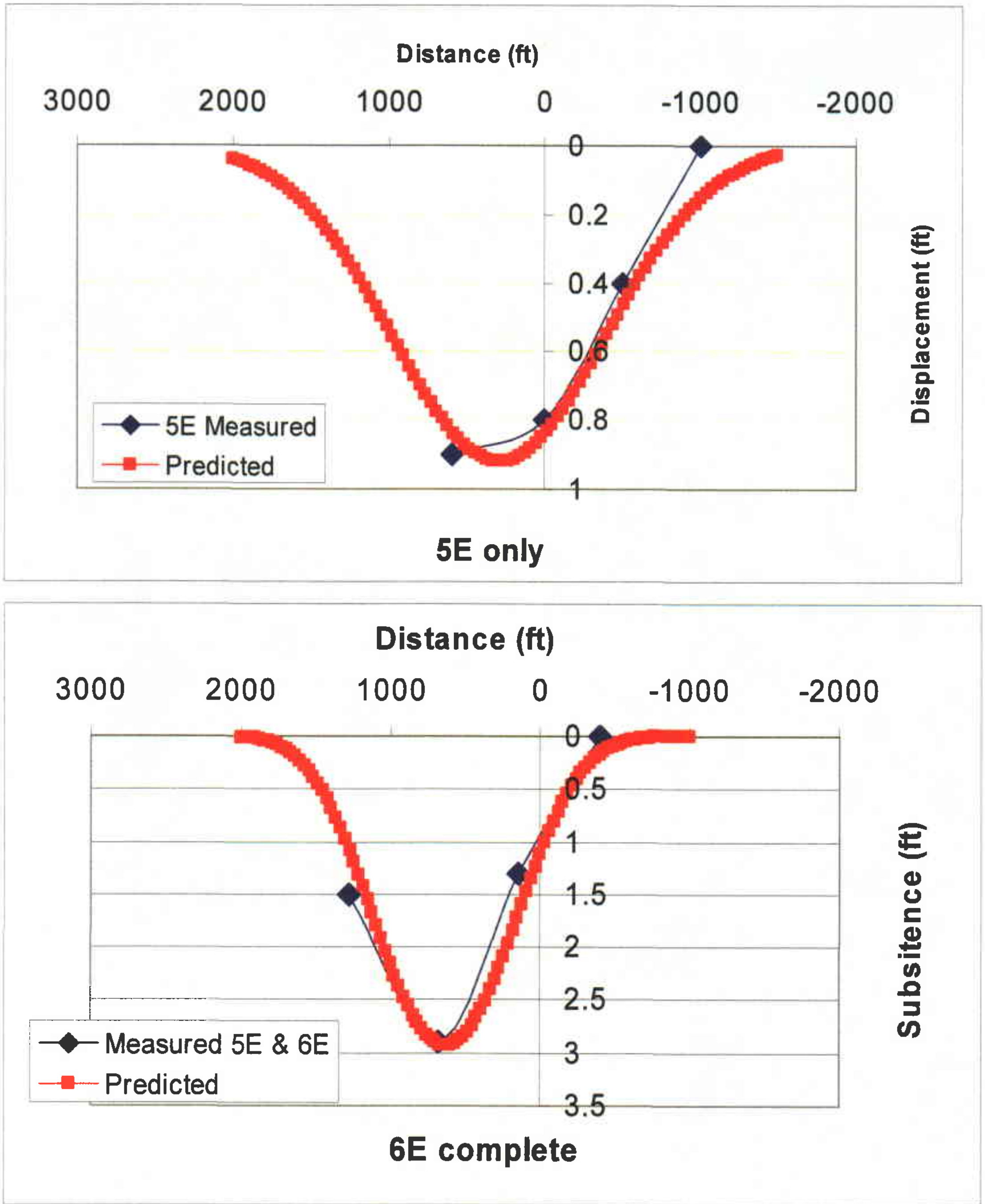


Figure 7. Compared measured and calculated subsidence after extraction of 5E and 6E panels.

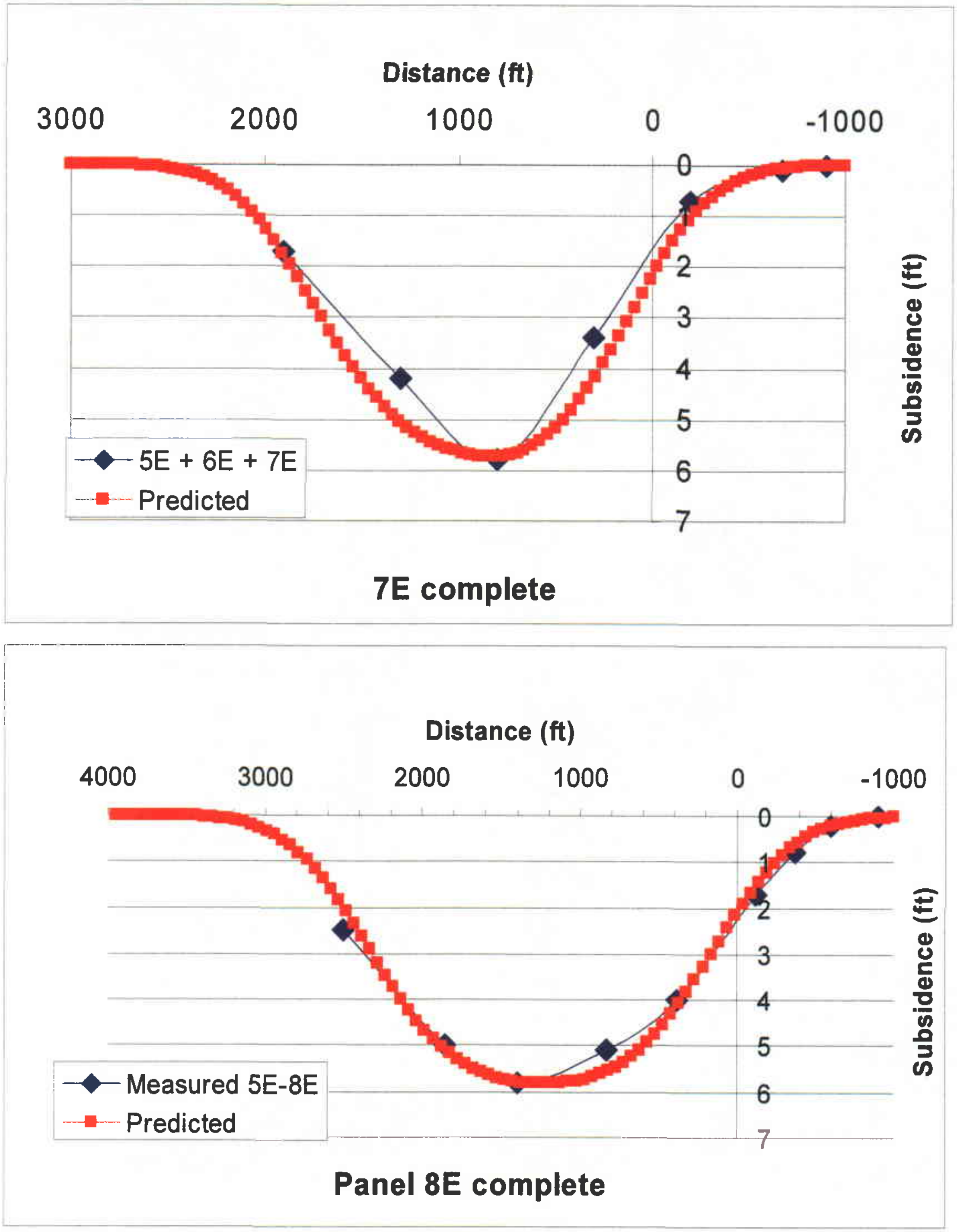


Figure 8. Compared measured and calculated subsidence after extraction of 7E and 8E panels.

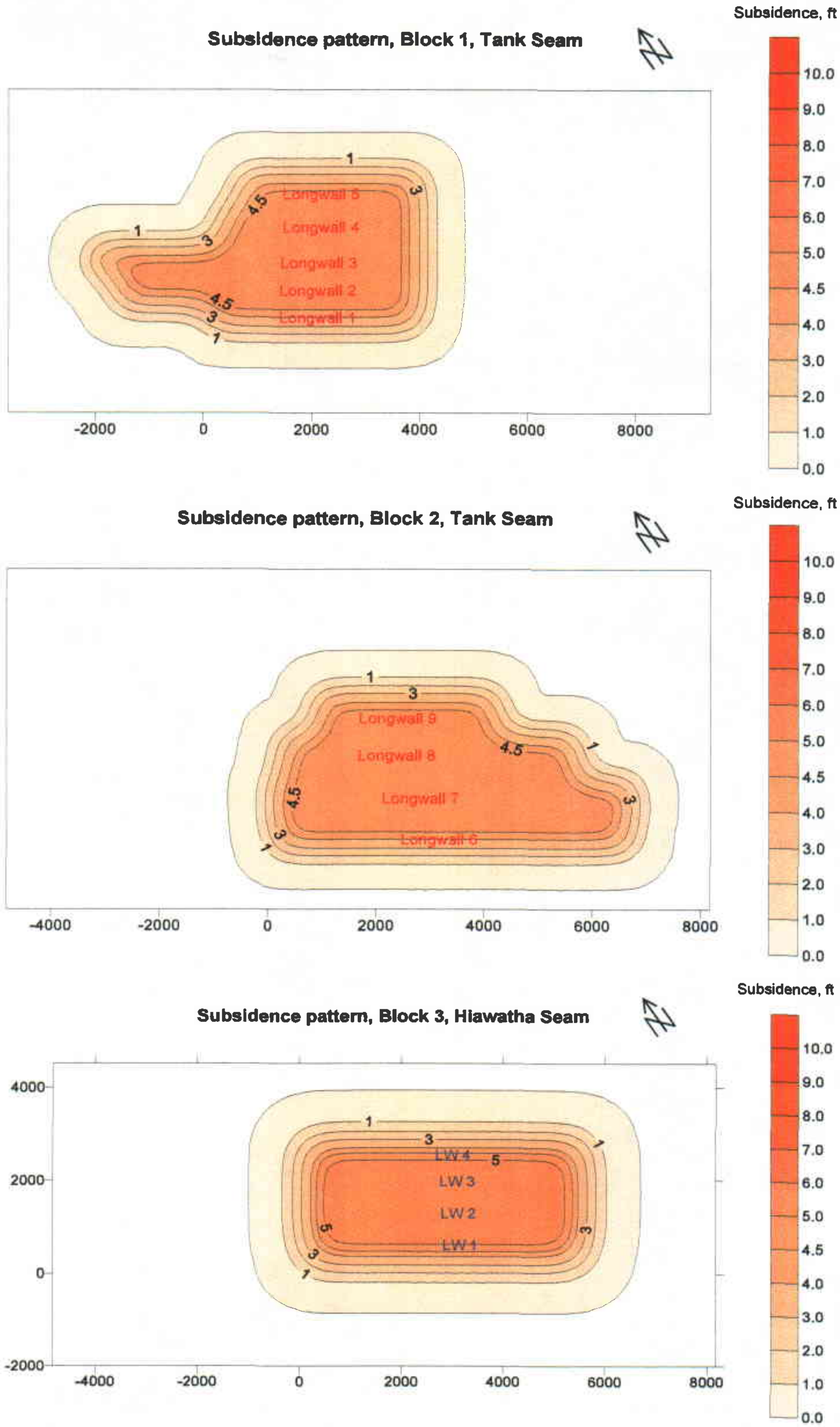


Figure 9. Subsidence pattern over longwall blocks 1, 2 and 3.

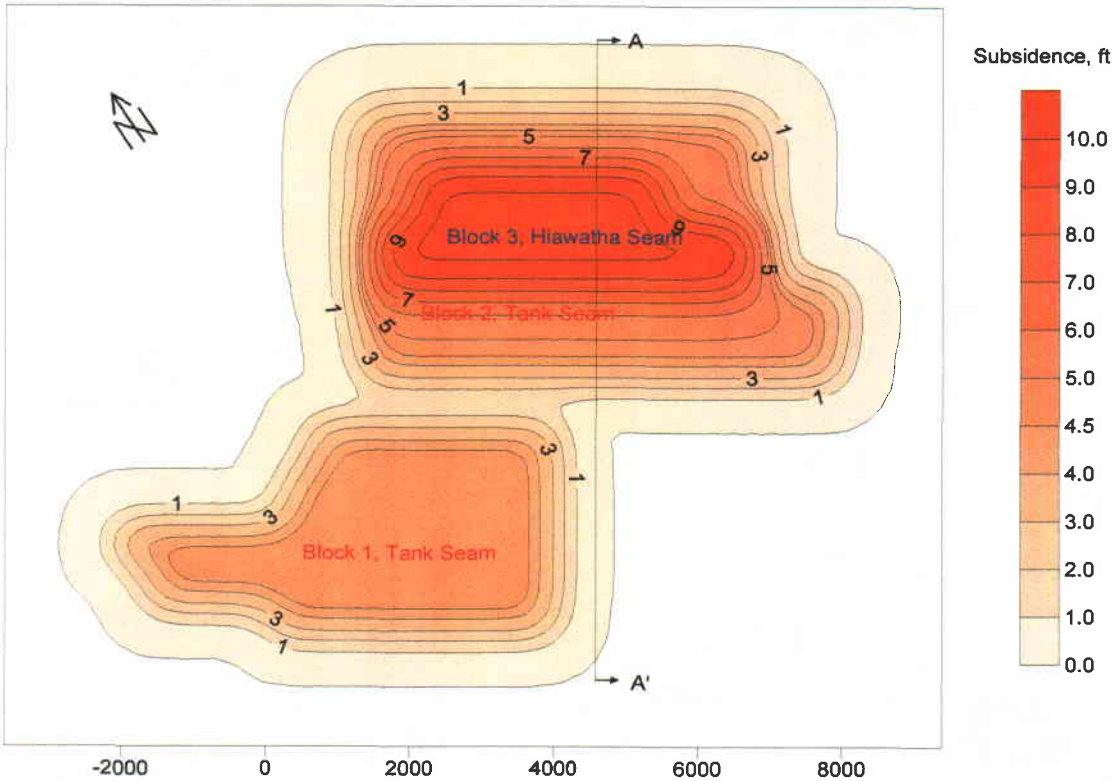


Figure 10. Total subsidence pattern after the extraction of blocks 1 through 3.

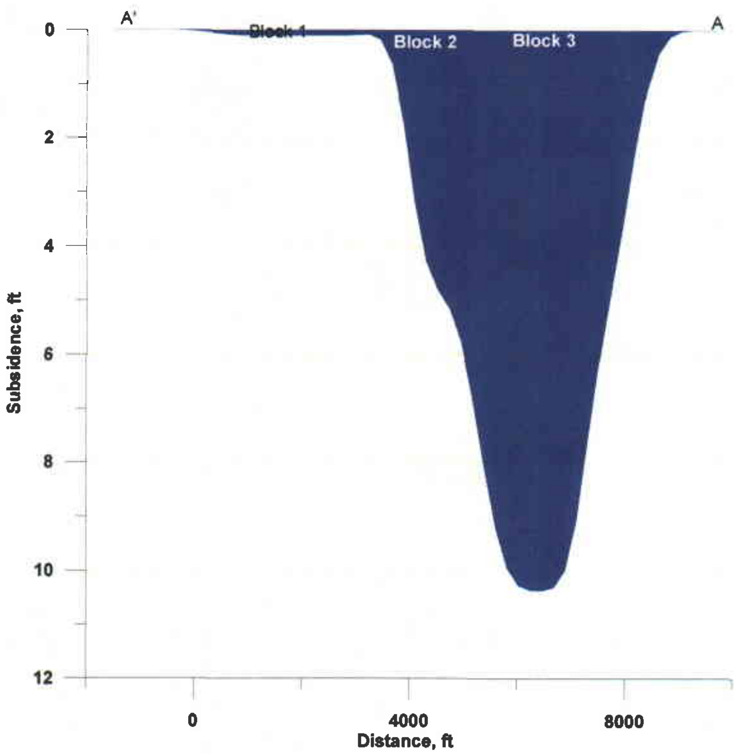


Figure 11. Typical subsidence profile at location A.

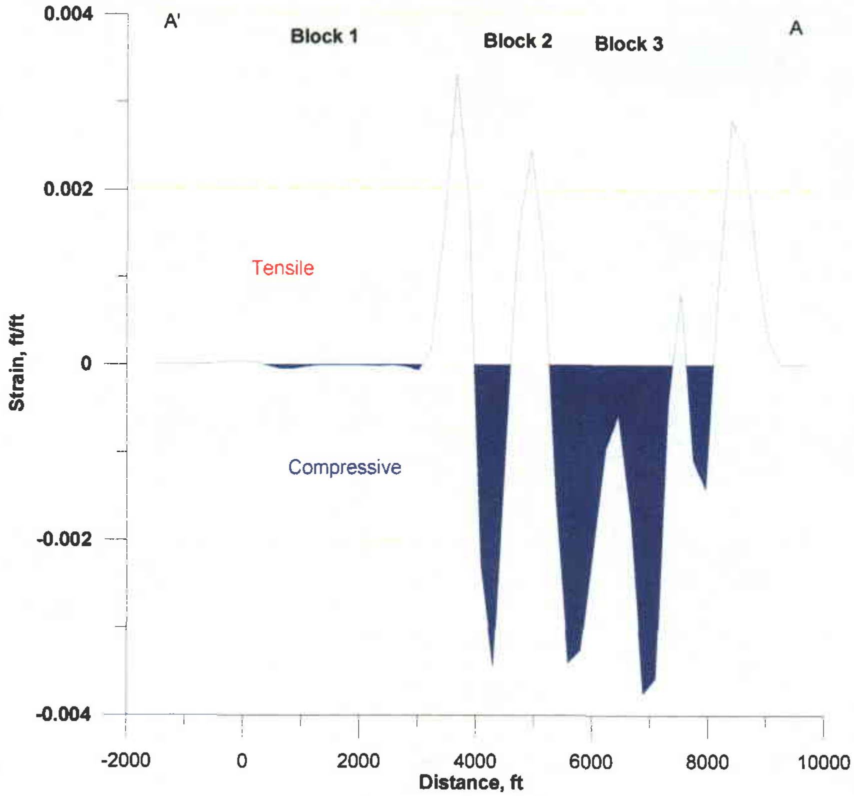


Figure 12. Typical final strain profile at location A.

5.0 MONITORING PROGRAM

A subsidence-monitoring program should be implemented to verify the subsidence predicted in this study and to record any mining-induced damage to surface resources.

Subsidence monuments should be monitored by surveying a monument line across the first longwall block. For verification purposes, it would be desirable to locate the monument line near the center of this block. The monument spacing of 50-ft is recommended over the first longwall panel for detailed comparison to the predictions. The monument spacing may be increased to 250-ft over panels 2 through 5.

From such monitoring, site-specific angle of draw, subsidence factor, and tensile strains can be calculated resulting in predictive subsidence techniques for the Bear Canyon study area. However, the arrangement and location of the monument line or method of survey can vary according to site-specific conditions influenced by topography, roads, etc.

Measurements should include a precision level survey to measure vertical settlement and possibly a steel tape extensometer to measure horizontal strain. GPS methods have recently become available and used in many western U.S. operations successfully. Alternatively, aerial photographic methods used extensively at the neighboring Trail Mountain and East Mountain, may be used.

C.W. Mining has not observed surface cracking above the existing Wild Horse Ridge panels and thus does not foresee the need for detailed monitoring. USBM researchers report very few mining-induced cracks over the East Mountain (Dyni 1991; Fejes 1986). MTI has designed and analyzed surface monitoring programs over Colorado mines (Maleki and others 2006). In some shallow mines, geologic staff conducts an annual crack survey over active longwall panel areas. A visual inspection is deemed sufficient over the deeper mines. The survey data include crack location, orientation, horizontal length, and width. Based on these measurements, MTI recommends a limited monitoring program so that the presence of surface cracks (if any) can be verified.

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Appendix 5-D

Toxic Materials & Handling

TOXIC MATERIALS AND HANDLING

Material that is contaminated with oil or grease or any other potentially acid or toxic matter, as determined by testing, will be placed against the highwall and covered with a min of 4 ft of non-acid and non-toxic forming fill material and reclaimed. Interim isolation of such material will be by use of berms created by a backhoe or loader.

Acid and toxic forming coal mine waste and material contaminated with coal, will be placed against the highwall and buried beneath a min of 4 ft of non-acid and non-toxic forming fill material during reclamation. See R645-301-540.

POTENTIAL HAZARDOUS WASTES

The following list includes the products which are used or may be used within the mine permit area, that are capable of producing hazardous wastes;

Diesel Fuel	Gasoline	Coal Oil
Carburetor Cleaners	Engine Degreaser	Windshield Washer Fluid
Lead Acid Batteries	Blasting Products	Solvents (Flash-point < 140°F)
Paints	Thinners	Dicers
Hydraulic Oil		

These products will be consumed in use or recycled to the extent possible to avoid costly disposal and company liability. Any of these or other materials that become potentially hazardous and their containers will be disposed of in the proper manner, at an approved disposal site.

COMMERCIAL TESTING & ENGINEERING CO.

GENERAL OFFICES: 228 NORTH LA SALLE STREET, CHICAGO, ILLINOIS 60601 · AREA CODE 312 726-8434

Reply to
Instrument Analysis Division
490 Orchard Street
Golden, CO 80401

Phone: 303-278-9521

November 23, 1981

Mr. Jack Blair
Commercial Testing & Engineering Co.
139 South Main Street
Helper, UT 84526

Co-op Mining Co.
Lab No. 57-7609

Re: IAD #97-H654-335-01

Analytical Report

One sample was received for analyses on October 28, 1981. This sample was given our identification IAD #97-H654-335-01.

A portion of the sample (≈100 g) was extracted at pH_5 for 24 hours according to the procedures of EPA/Test Methods for Evaluating Solid Wastes, SW-846, 1980, EP Toxicity. The sample required ≈170 mls of 0.5 N acetic acid to adjust the pH to 5. The extracted solution was brought to volume (2000 ml) and filtered with a 0.45 μm membrane filter. A portion of the filtered extract solution was acidified with nitric acid prior to metals analyses.

The solution was analyzed for Lead, Silver, Barium, Cadmium, and Chromium by flame atomic absorption; for Arsenic and Selenium by hydride generation atomic absorption; and for Mercury by cold vapor flameless atomic absorption using a permanganate/persulfate digestion and the gold amalgamation analytical technique to concentrate the Mercury.

The results of these determinations are presented in Table No. I and are reported in milligrams per litre (mg/L) in the filtered extract solution. The EP Toxic maximum contaminant levels are also presented.

Charter Member

Table No. I
(mg/L)

EP Toxicity

Parameter	Co-Op Mining 57-7609	Maximum Contaminant Levels
Arsenic	$\leq$0.001	5.0
Selenium	$\leq$0.001	1.0
Mercury	0.0004	0.2
Silver	$\leq$0.005	5.0
Barium	1.1	100
Chromium	$\leq$0.01	5.0
Cadmium	$\leq$0.005	1.0
Lead	$\leq$0.05	5.0

If there are any questions concerning these results, please call.

Bruce A. Hale
Section Supervisor

M. L. Jacobs
M. L. Jacobs, Ph.D., Mngr.
Instrumental Analysis Div.

BAH/cl

COMMERCIAL TESTING & ENGINEERING CO.

July 22, 1983

Reply to
Instrumental Analysis Division
10 Orchard Street
Golden, CO 80401

Phone 303-278-9521

Mr. Jack Blair
CT & E Co.
224 South Carbon Ave.
Price, Utah 84501

CO-OP MINING COMPANY
Pond Sample
Lab No. 57-13312

Re: IAD # 97-M179-335-01

Analytical Report

One coal sample was received for analysis on July 12, 1983. This sample was assigned our IAD identification # 97-M179-335-01.

The sample was prepared to No.10 mesh size in accordance with the procedure of U.S.D.A. Handbook #60. Electrical Conductivity and pH were determined in accordance with the same publication.

Acid/Base Potential was determined in accordance with the procedure of the Environmental Protection Agency, EPA-670/2-74-070. This procedure is used for the Wyoming DEQ and in telephone conversation with the Utah Division of Oil, Gas & Mining we were advised that this procedure is acceptable for the requirements of the State of Utah.

The results of these determinations are presented in Table No.1 and are reported in units as indicated in the Table.

Table No. 1

Parameter	57-13312
pH, paste (Standard Units)	7.6
Electrical Conductivity (μmhos/cm)	195
Acidity Potential*	0
Neutralization Potential*	29.8
Acid/Base Potential*	29.8

*Values are reported in Tons $CaCO_3$ Equivalent / 1000 Tons.

Texture determination was not performed as the sample is carbolithic and thus the determination of Sand, Silt and Clay fractions is not applicable in this case.

If you have any questions concerning these results, please call.

Harold A. Connell
Assistant Lab Manager

R.L. Taylor, Ph.D., Manager 2-2 7-2-93
Instrumental Analysis Division

Table II

Parameter	Roof	Coal	Floor
Acid Potential, tons $CaCO_3$/1000 tons	1	15	<1
Neutralization Potential, tons $CaCO_3$/1000 tons	595	6.3	488
Clay Content, Wt. %	---	---	25.4

If you have any questions concerning these results, please feel free to call.

Martha L. Turner
Martha L. Turner
Supervisor
Environmental Section

MLT/vmc

64

COMMERCIAL TESTING & ENGINEERING CO.

GENERAL OFFICES: 1919 SOUTH HIGHLAND AVE., SUITE 210-B, LOMBARD, ILLINOIS 60148 • (312) 953-9300

PLEASE ADDRESS ALL CORRESPONDENCE TO
224 SO. CARBON AVE., PRICE UT 84501
OFFICE TEL. (801) 637-7540

[redacted]

July 7, 1986

Co-op Mining Co.
P.O. Box 300
Huntington, Utah 84528

Sample [illegible]
by Mel Coonrod

Roof Sample
Coal Sample
Floor Sample

Kind of sample reported to us: Soil

Sample taken at: Hiawatha Seam

Sample taken by: Mel Coonrod

Date sampled: 6-1-86

Date received: 6-2-86

Analysis report no. 57-21437,38,39

SULFUR FORMS

	Roof	Coal	Floor
Pyritic Sulfur	0.06	0.01	0.02
Sulfate Sulfur	0.01	0.01	0.02
Organic Sulfur (Diff.)	xxxx	0.56	xxxx
Total Sulfur	xxxx	0.58	xxxx

Reported As Dry Basis Only

COMMERCIAL TESTING & ENGINEERING CO.
GENERAL OFFICES: 1919 SOUTH HIGHLAND AVE., SUITE 210-B, LOMBARD, ILLINOIS 60148 • (312) 953-9300

Member of the SGS Group (Société Générale de Surveillance)

PLEASE ADDRESS ALL CORRESPONDENCE TO
490 ORCHARD ST., GOLDEN, CO 8040[illegible]
TELEPHONE (303) 278-95[illegible]

Commercial Testing & Engineering
224 So. Carbon Avenue
Price, Utah 84501

Date: July 03, 1986
IAD #97-W406-335-03
Received: 06/18/86

Material: Soil

Procedure: EP Toxicity per EPA, Hazardous Waste and Consolidated Permit Regulations, Federal Register, Monday, May 19, 1980.
Acid Potential, Neutralization Potential and Clay Content per EPA, 600/2-78-054.

Results: EP Toxicity reported in milligrams per Liter (mg/L), on an extract basis.
Acid Potential and Neutralization Potential reported as tons $CaCO_3$/1000 tons material.
Clay Content reported in weight percent (wt. %).

Table I
EP Toxicity

Parameter	Roof	Coal	Floor
Arsenic, As	<0.001	<0.001	<0.001
Barium, Ba	<0.8	<0.8	<0.8
Cadmium, Cd	<0.006	<0.006	<0.006
Chromium, Cr	<0.02	<0.02	<0.02
Lead, Pb	<0.04	<0.04	<0.04
Mercury, Hg	<0.0002	<0.0002	<0.0002
Selenium, Se	<0.002	<0.002	<0.002
Silver, Ag	0.017	<0.008	0.019
inital pH, s.u.	9.3	7.3	9.5
final pH, s.u.	6.9	4.9	5.1
mLs acetic acid added, per 100g sample	400	50	400

Inter-Mountain Laboratories, Inc.

2506 West Main Street | Farmington, New Mexico 87401 | Tel. (505) 326-4737

CO - OP MINING COMPANY
HUNNINGTON, UTAH

Date Reported: March 7, 1989

Lab No.	Location	Depths	pH s.u.	EC mmhos/cm @ 25C	Saturation %	Calcium meq/l	Magnesium meq/l	Sodium meq/l	SAR	Sand %	Silt %	Clay %	Texture	Organic Carbon %
3911	SED.POND "A" 1	0.0-0.0	7.9	1.00	36.6	4.46	2.93	4.14	2.15	90.9	9.1	0.0	SAND	8.78
3912	SED.POND "A" 2	0.0-0.0	7.7	2.31	32.2	16.9	9.68	4.06	1.11	89.1	10.7	0.2	SAND	9.22
3913	SED.POND "A" 3	0.0-0.0	8.7	1.18	37.1	7.59	3.30	1.52	0.65	89.3	10.7	0.0	SAND	9.33
3914	SED.POND "A" 4	0.0-0.0	7.9	1.26	38.4	6.20	4.52	1.75	0.76	80.3	19.7	0.0	LOAMY SAND	8.86
3915	SED.POND "A" 5	0.0-0.0	8.0	0.90	36.8	4.11	3.05	1.90	1.00	80.9	19.1	0.0	LOAMY SAND	9.12

Miscellaneous Abbreviations: SAR= Sodium Adsorption Ratio, CEC= Cation Exchange Capacity, ESP= Exchangeable Sodium Percentage, Exch= Exchangeable, Avail= Available

Inter-Mountain Laboratories, Inc.

2506 West Main Street | Farmington, New Mexico 87401 | Tel. (505) 326-4737

CO - OP MINING COMPANY
HUNNINGTON, UTAH

Date Reported: March 7, 1989

Lab No.	Location	Depths	Total Sulfur %	T.S. ABP t/1000t	Pyritic Sulfur %	Organic Sulfur %	NP	ABP	AB	Boron ppm	Selenium ppm
3911	SED.POND "A" 1	0.0-0.0	0.38	160.	0.01	0.35	171.80	159.93	11.87	0.28	0.02
3912	SED.POND "A" 2	0.0-0.0	0.34	176.	0.03	0.30	186.42	175.80	10.62	0.47	<0.02
3913	SED.POND "A" 3	0.0-0.0	0.34	160.	<0.01	0.34	171.01	160.39	10.62	0.40	<0.02
3914	SED.POND "A" 4	0.0-0.0	0.35	168.	<0.01	0.35	179.17	168.23	10.93	0.49	0.02
3915	SED.POND "A" 5	0.0-0.0	0.34	158.	0.01	0.33	169.08	158.45	10.62	0.39	<0.02

Abbreviations used in acid base accounting: T.S.= Total Sulfur, AB= Acid Base, ABP= Acid Base Potential, PyrS= Pyritic Sulfur, Pyr+Org= Pyritic Sulfur + Organic Sulfur, Neut. Pot.= Neutralization Potential

Appendix 5-E

Surface Blasting Plan

GENERAL

This blasting plan is designed to meet the requirements of R645-301-524. Occasionally during surface site preparation, surface blasting may be required to break rocks too large to move with equipment. The following sections describe the procedures and limitations, which will be adhered to in these blasts.

Blaster Certification

Co-Op Mining company currently employs a certified blaster who has been trained and certified by the Utah Division of Oil, Gas & Mining in accordance with R645-105. A copy of the blaster certificate is maintained on file at the minesite.

The certified blaster will be present during all surface-blasting activities, and will ensure that all state and federal regulations are adhered to during blasting. The blaster and at least one other person will be present at the firing of a blast.

Blast Design

The following blast design pertains to blasting within 1,000 feet of any building used as a dwelling, public building, school, church, or community or institutional building outside the permit area, or within 500 feet of an active or abandoned underground mine. Approval will be obtained from MSHA prior to conducting blasting within 500 feet of an active underground mine. There are no underground mines within 500 feet of the Wild horse Ridge potential blasting areas. This design describes the use of blasting to break large rocks encountered during construction. This design will be used in the construction of the Wild Horse Ridge conveyor access roads in the event rocks are encountered which are too large for the equipment to move, or cutting into bedrock is required. A hunting cabin exists within the permit area approximately 750 feet from the nearest potential blasting zone for the Wild Horse Ridge areas. This building is not used as a dwelling, public building, school, church, or community or institutional building. No other buildings exist outside the permit area within 1,000 feet of the proposed Wild Horse Ridge roads and facilities.

Two types of blasting are anticipated during normal operation and construction, special one time blasting designs will be included as attachments. The first is a situation where large rocks will require breaking up to move. The second type is bench blasting in areas where bedrock is encountered which must be cut into. In both cases, holes to be used will be 1½" diameter, 4' deep and will be spaced 4' apart. Figure 5E-1 shows the typical pattern and the layout for the bench holes. To break up rocks, holes will be drilled into the rock at an adequate depth to fracture the rock, not to exceed 4'. Holes will be spaced approximately 4' apart.

The explosive to be used is Irecoal D 378 permissible explosive, an explosive which uses a non-nitroglycerin base gel dynamite. This explosive is specifically designed for use in drillhole patterns, especially when shooting off solid. Ireco electric 25 ms delay detonators will be used to delay the holes in a pattern which will contain the shot within the cut area created by the previous shot. 1.3 pounds of explosive per each hole will be used, providing a powder factor of 1.2. The following standards will be maintained for all blasts.

Airblast Limits and Ground Vibration (R645-301-524.600). Airblast and ground vibration limits at the location of any building structure outside the permit area will not exceed those outlined in R645-301-524.621 and R645-301-524.642. In an effort to minimize airblast and fly rock, a satchel-type explosive and mud-packing will not be used.

Pre-Blast Survey

Since each blast will not exceed 5 pounds of explosive weight, the requirements of a pre-blast survey are not applicable to the blasting plan. During the construction of the #4 Mine portals blasting will exceed 5 pounds, this is described in attachment A.

Blasting Schedule

Since each blast will not exceed 5 pounds of explosive weight, a blasting schedule and public notification will not be required.

Warning Signs, Signals and Access Control

During periods of construction involving blasting, warning signs will be posted as described in R645-301-524.

Warning and all clear signals of different patterns, audible within a range of one-half mile from the point of the blast, will be given to inform anyone in the area of the status of blasting activities. The signals and their patterns and meanings will be described in the "Warning" sign at the entrance to the property. Each person within the permit area, or anyone who resides or regularly works within one-half mile of the permit area will be notified of the meaning of the signals.

Access to the area will be controlled to prevent livestock or unauthorized persons from entering the area by individuals monitoring the perimeter of the blasting area until the all-clear signal is given.

Blasting Record

A blasting record will be maintained onsite of each blast performed, and will be made available to the Division and the public upon request. The record will include the following:

Name of the operator conducting the blast

Location, date and time of the blast

Name, signature and certification number of the blaster

Identification, direction and distance from the nearest blast hole to the nearest building outside the permit area

Weather conditions

A record of the blast, as required in R645-301-524.740.

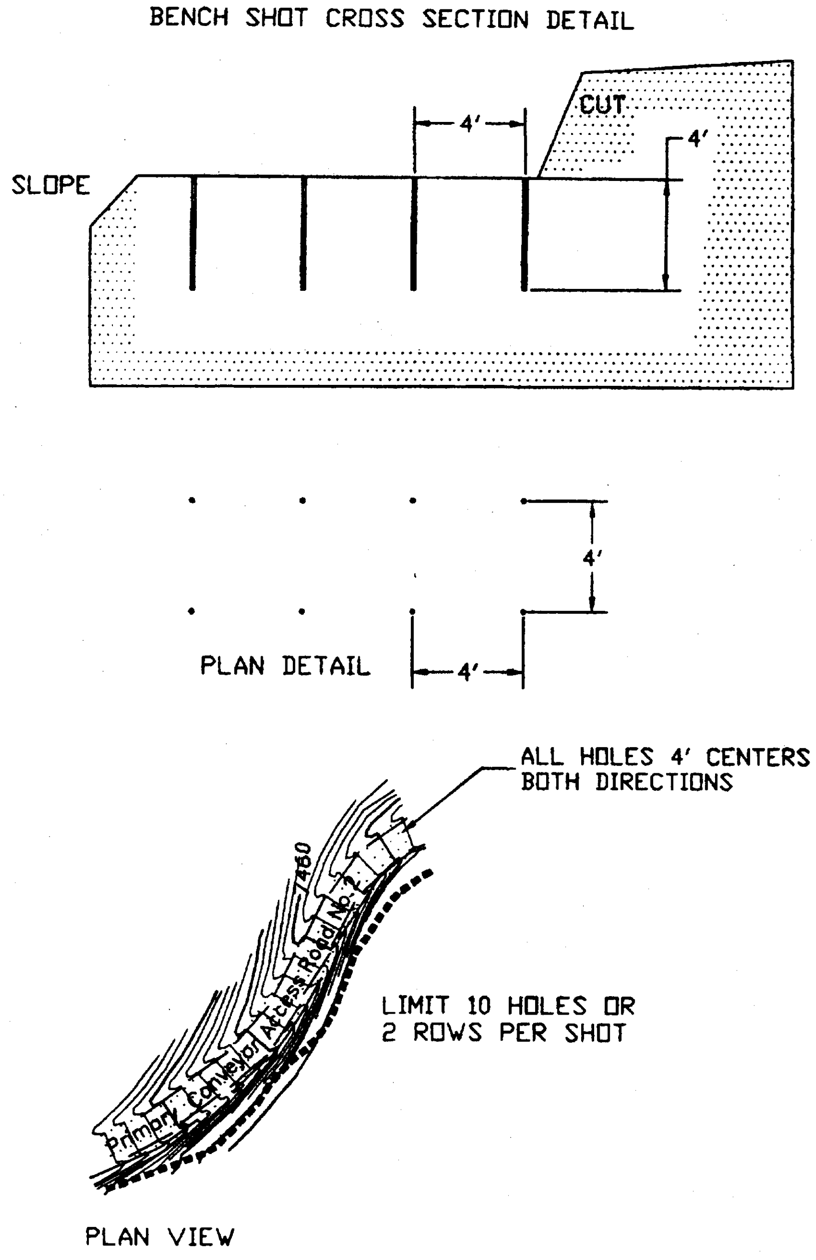
Blaster Certification

I hereby certify that this blasting plan as been prepared under my supervision in accordance with R645-301-524.

10/22/2002

Signature of Certified Blaster Date

Figure 5E-1
Drillhole Pattern



ATTACHMENT A
#4 MINE ROCK TUNNELS

This attachment has been included to address the blasting that will occur during the development of the WHR #4 mine portals. All blasting procedures outlined in Apendix 3M will be followed except for the changes that have been outlined below.

Blast Design

Holes will be drilled 10 feet deep . Figures 1,2, and 3 show the typical patterns and the layout for the holes. Eight 0.78lb 1 1/4" diameter by 12" long sticks will be placed in each hole. The explosive to be used is Irecogel B permissible explosive, an explosive which uses a non-nitroglycerin base gel dynamite. Coaldet, Millidet or equivalent electric millisecond delay detonators will be used to delay the holes in the patterns shown. A scorpion HCCR (MSHA LCBU-1) shot firing unit will be used to initiate the detonators.

Pre-Blast Survey

Only two dwellings are located within one-half mile of the blast area. One is the Hunting Cabin owned by Sportsmans and a residential home owned by Kenny Defa the mine superintendent. Both owners have been notified in writing of the blast. A pre-blast survey of both buildings has been conducted and his held on file at the mine site.

Blasting Schedule

Blasting will be conducted between sunrise and sunset. Blasting will generally take place Monday thru Friday, but may include Saturday and Sunday if needed. A detailed time table will be given to local governments, and residents within one-half mile, and will also be published in the local newspaper at least ten days before blasting begins. If the schedule changes, local governments, residents, and the local paper will be notified. The blasting schedule will contain all information required under R645-301-524.460.

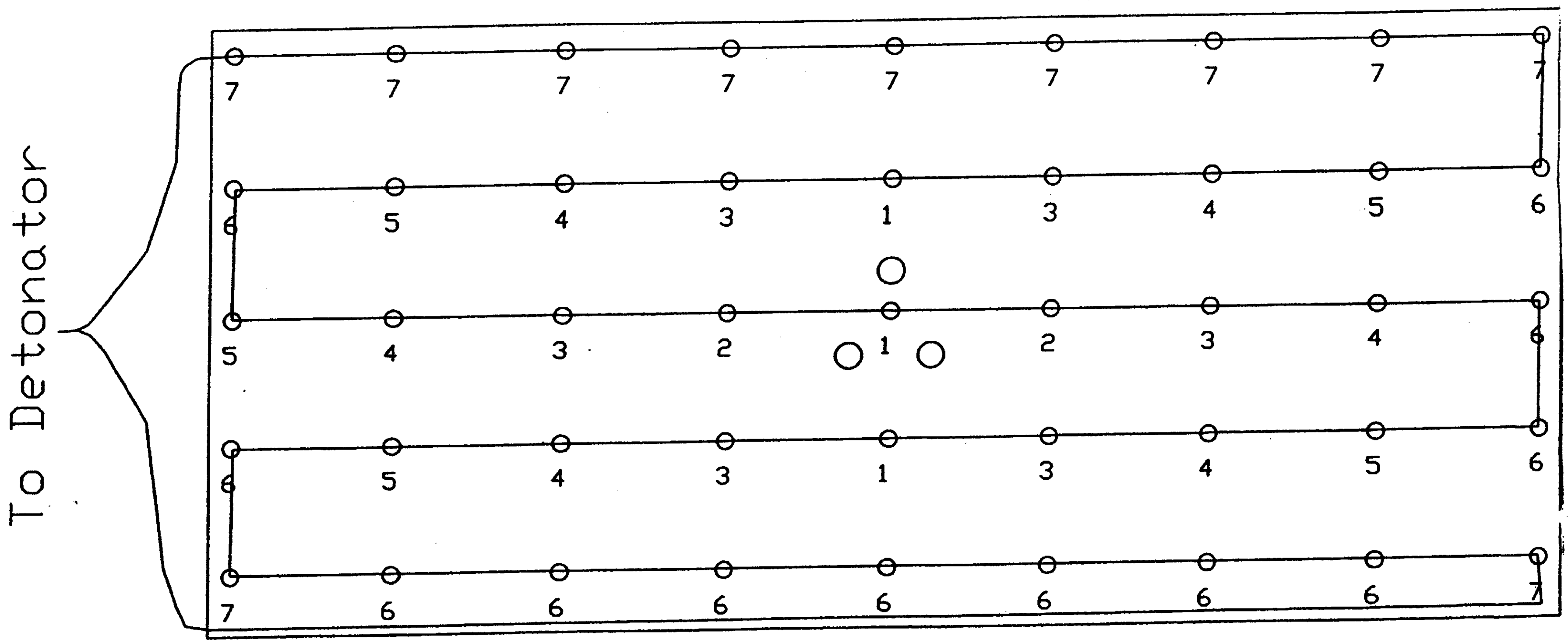
I hereby certify that this blasting plan as been prepared under my supervision in accordance with R645-301-524.

10/16/2003

Signature of Certified Blaster | Date

10/13/03

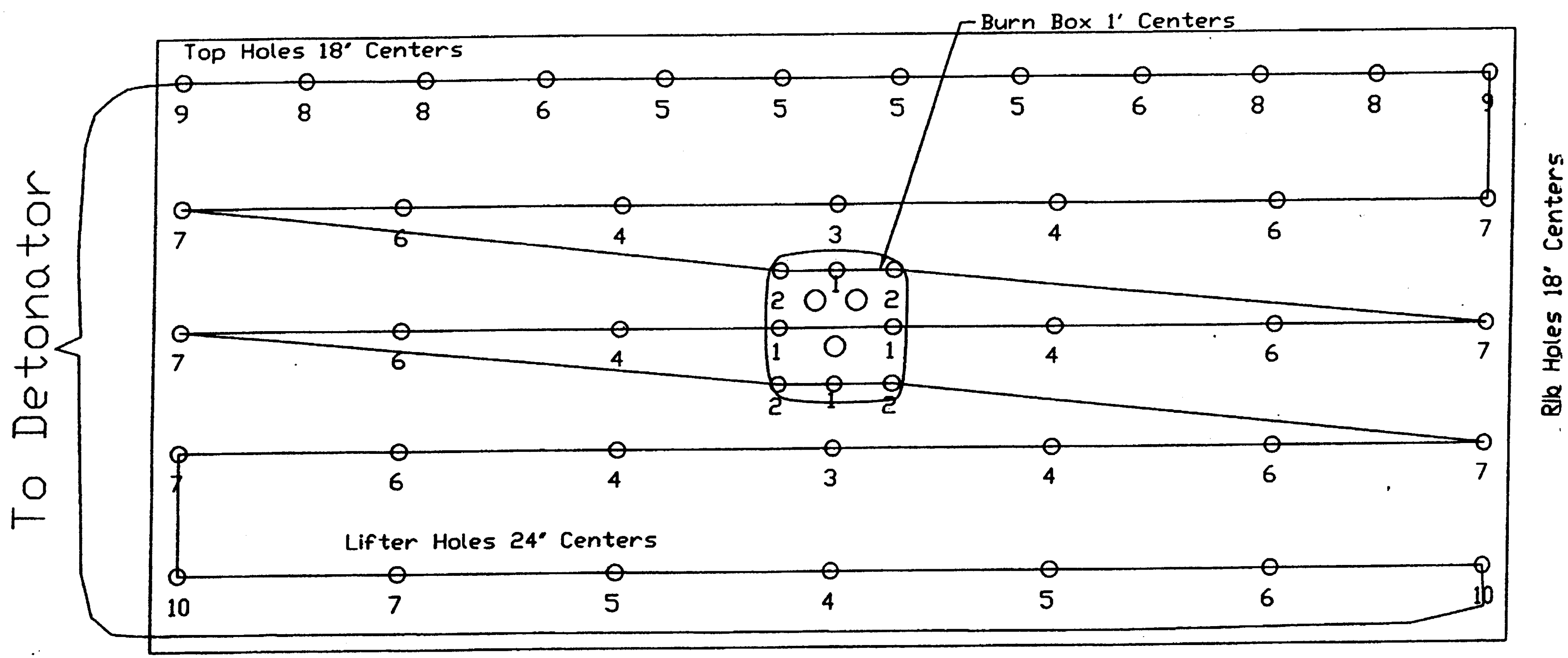
Figure 1



Delay pattern may vary based on conditions
Minimum spacing in coal 2.4"
Minimum spacing in rock 18"

B.C.

Figure 2

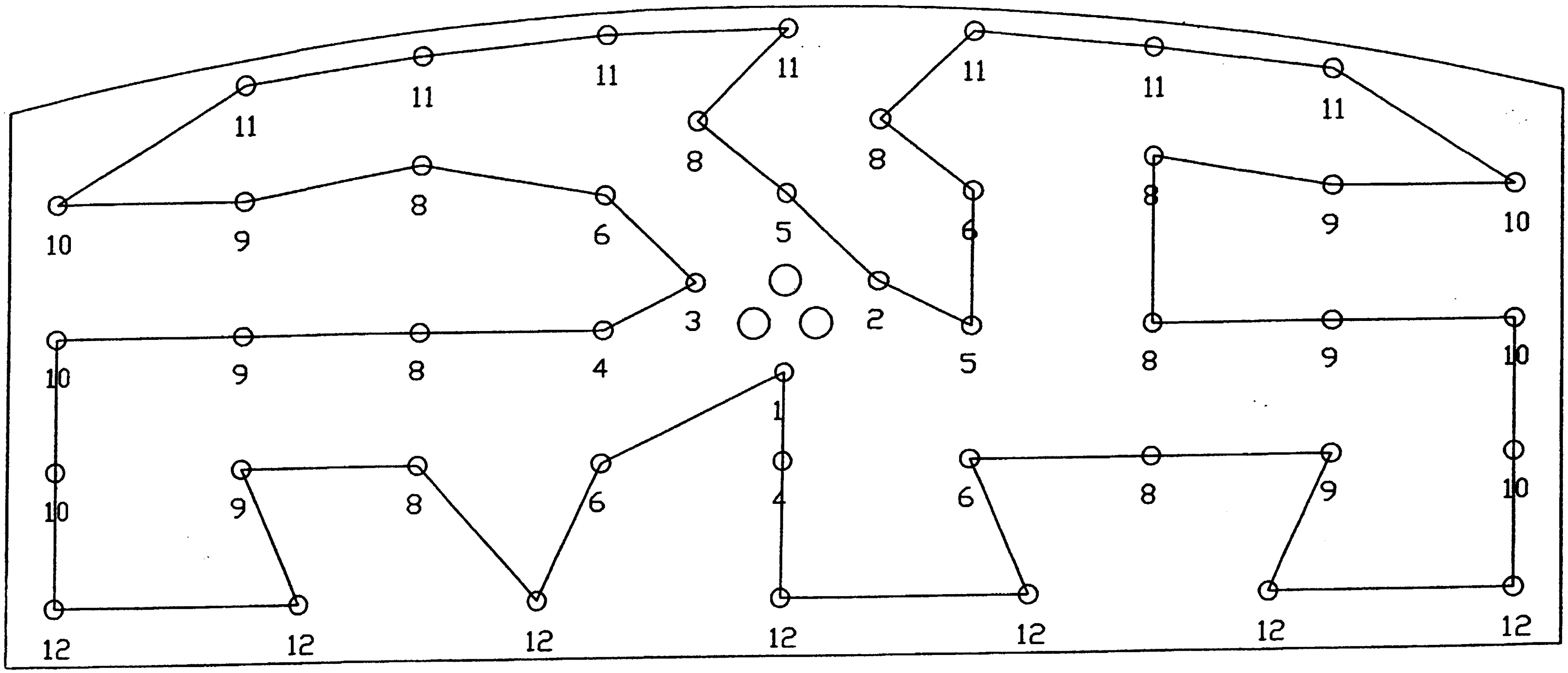


Delay pattern may vary based on conditions
Minimum spacing in coal 2.4"
Minimum spacing in rock 18"

B.C.

10/13/03

Figure 3



Delay pattern may vary based on conditions
Minimum spacing in coal 2.4"
Minimum spacing in rock 18"

B.C.

Appendix 5-F

Mine Roads

GENERAL COMMENTS

Construction

All roads shall be constructed and maintained in such a manner that the approved design standards are met throughout the life of the entire transportation facility. This shall include maintenance of the surface, shoulders, parking, side areas, and erosion control structures for safe and efficient utilization of the road. Road are shown on Plates 5-2. Cross sections and profiles are found on Plates 5-4.

The horizontal alignment of each road is consistent with the existing topography and with the volume, speed, and weight of anticipated traffic. The highly traveled Primary Haul Road and Portal Access Road are surfaced with 4 in. min of durable road base material. The high percentage of coarse granular material in the native soil provides for adequate surfacing of the remaining roads. Additional road base may be added to all roads as required. Damage to the roads from use or weather events shall be promptly repaired.

Ditches and culverts have been designed and installed to control and safely pass or contain run-off from a 10-yr, 24-hr precipitation event. Ditches are rip-rapped as required. Culverts are fitted with trash racks to prevent plugging and buried adequately to prevent crushing. Rock or concrete headwalls are provided at inlets to all culverts, and rip-rap or other erosion protection is provided at the outlet. Design and description of drainage structures can be found in Chapter 7. Ditches and culvert locations are shown on Plates 7-1.

Environmental Protection

All roads shall be constructed and maintained in such a manner to prevent damage to fish, wildlife, and related environmental values. This is accomplished by;

a. Maintaining hydrologic controls, such as ditches, culverts, diversions and sedimentation ponds to assure that disturbed drainage is conveyed away from undisturbed drainage and either held or cleaned before releases.

b. Watering of roads as necessary to reduce fugitive dust.

c. Protection of wildlife within the permit area and reporting of sightings of threatened and endangered species.

d. Contemporaneous reclamation.

e. Advocating good-housekeeping practices to reduce the possibility of contamination of surface waters in the area.

f. Co-Op is committed that all support facilities will be restored to prevent damage to fish, wildlife, and related environmental values and the possibility of additional contributions of suspended solids to stream flow or runoff outside the permit area will be minimal.

Reclamation

All roads shall be removed upon completion of the mining operation except those approved as part of the post-mining land use. The timing and procedure of removal and reclamation is discussed in detail under the Backfilling and Grading Plan in R645-301-553 See R645-301-540, and Chapters 7, 2 and 3 for full reclamation procedures.

During reclamation road surfacing material will be removed and salvaged or buried as fill material in the reclamation of highwalls, see Appendix 5-I. Reclamation will then be accomplished by ripping up the remaining base, and ensuring that suitable plant growth material is in place prior to planting the area with the approved seed mix. During this time, all culverts shall be removed and either salvaged or disposed of in an approved landfill, and the natural drainage patterns shall be restored.

PRIMARY ROADS

There are 11 Primary roads within the Permit Area. Each road is described below. Construction of the Tank Seam Access road is described in Appendix 5-G. Construction of the Wild Horse Ridge Access road and Conveyor Access roads are described in Appendix 5-J. Construction of the No. 4 Mine Access Road is described in Appendix 5-K.

Primary Haul Road

The Bear Canyon Haul Road is approx 3690 ft long from the gate to the base of the Portal Access Road and Tipple Access Road. See Plates 5-2 and 5-4. As shown on Plates 5-2, this portion of the road is included in the Permit Area. This Primary road is constructed approximately 30 ft wide and is surface with 4 in. minimum of road base material. Approximately 160 ft of the road adjacent to the scalehouse is to be surfaced with 6 in. of pavement. Installation of the black top will involve hauling in a hot mix from a local asphalt plant and laying it on the road. The road is crowned in the middle as shown on the cross section, Plate 5-4.

Reclamation of this portion of the Bear Canyon Road will occur at approx the same time as the final removal of the sedimentation ponds and diversions on the mine site. Removal of the asphalt on this road will consist of hauling the material to a landfill approved for solid waste disposal and/or recycling of the material in cooperation with a local asphalt plant. The road will be narrowed, but will remain in place for the post-mining land use.

Primary Access Roads

The Portal Access Road is approx 3,170 ft long. The road was originally constructed for access to the old Bear Canyon Mine, and has since been widened and fitted with proper drainage controls to protect the environment. The road is designed, used and maintained to meet the DOGM requirements, and to control or minimize erosion and siltation, air and water pollution, and damage to public or private property.

The road is located along the canyon floor above the stream, and along the stable slope leading to the portals. The overall grade of the road does not exceed l:V:10h (10 pct) and the max grade does not exceed l:V:6.5h (15 pct).

As mentioned earlier, the initial road was constructed under pre-law conditions, using the cut and fill side-cast method. A stability analyses was performed on the road by Dames & Moore in 1981 (Appendix 5-H). Their conclusion was that the Bear Canyon Portal Access Road has a stability factor of safety of a minimum of 1.43, and ranges upward to 2.15.

Shower House Road

This road is 200 ft long and is to be constructed in 1993 to provide access to the shower house area. The road varies from a 0% to 11% grade, with an average grade of 7%. The embankment of the road where it crosses Bear Creek will be constructed at a maximum slope of 1V:1.5H, or 67%. The properties of the material to be used are similar to the material of existing embankments. Appendix 5-H contains a slope stability analysis of similar material on nearby embankments. This analysis shows a minimum slope stability safety factor of 1.68, assuming saturated conditions, which exceeds the minimum required safety factor of 1.3 (pg 5-H-18 thru 5-H-20).

Road to Sediment Pond A

This road is 340 ft long and was constructed to allow access to the Sediment Pond and to facilitate cleaning of the drainage to the pond. The road is in actuality the disturbed drainage ditch, D-6D, to Sediment Pond A, and is used infrequently to clean sediment from the pond. The road has an overall slope of approx 3.5 pct and does not exceed 10 pct at any point.

Tipple Access Road

This road is 600 ft long, and was constructed to provide access to the Coal Preparation Facility. The road has an overall slope of approx 12 pct, and does not exceed 25 pct at any point.

Shop Road

This road is 160 ft long, and provides access to the Shop Pad. The road has an overall slope of approximately 1 pct, and does not exceed 5 pct at any point.

Tank Seam Access Road

This road is approximately 3,150 ft long, and provides access to the Bear Canyon #2 Mine, located in the Tank Seam. The road has an overall slope of approximately 9 percent, and does not exceed 17 percent at any point. Construction of this road is discussed in Appendix 5-G.

The Tank Seam Access Road will be maintained in accordance with the requirements of this Appendix. During snow storms, snow will be plowed to and stored against the cut slope of the road along the ditches, in order to prevent saturation of the fill outslopes along the road due to snow melt

Wild Horse Ridge Access Road

This road is approximately 4,850 ft long, and provides access to the Bear Canyon #3 Mine, located in the Blind Canyon Seam in Wild Horse Ridge. The road has an overall grade of 10.5 percent, and does not exceed 18 percent at any point. This road existed prior to mining and will remain in place to meet the post-mining land use. Construction of this road is discussed in Appendix 5-J.

Wild Horse Ridge Conveyor Access Roads

These two roads provide access to remote portions of the Wild Horse Ridge conveyor. The lower road (No. 1) is approximately 600 ft long, averaging 10 percent grade. The upper road (No. 2) is approximately 590 ft long, averaging 19.5 percent grade. Construction of these roads are described in Appendix 5-J.

These roads will be reclaimed in the same manner as the Tank Seam Access Road, as described in R645-301-540.

Wild Horse Ridge No. 4 Mine Access Road.

This road is approximately 2,000 ft, long, averaging 10 percent grade. Part of this road existed prior to mining and will remain in place to meet the post-mining land use. Construction of this road is described in Appendix 5-K.

ANCILLARY ROADS

The only Ancillary Road within the permit area is a jeep trail that was constructed pre-law, probably as a cattle trail. This road is shown on Plate 5-2C. The road is blocked off, is not within the disturbed area, and is not used; therefore, no maintenance or reclamation plan is proposed for this trail.

CO-OP MINING CO.
BOX 1245
HUNTINGTON, UTAH 84528

Aug. 8, 1983

S U B M I T T A L

RECEIVED
AUG 08 1983
DIVISION OF
OIL GAS & MINING

TO:

DIVISION OF OIL GAS AND MINING
4241 STATE OFFICE BUILDING
SALT LAKE CITY, UTAH 84114

Co-op Mining Co. submits the following plans for that portion of the haul road in the Bear Creek Canyon designated as a private road in the enclosed copy of the aggreement between Emery County and Co-op Mining Co. The road coincides with the existing Bear Canyon road, which follows the original contour of the land, minimizing additional disturbance or adverse effects on the environment. There will be no cut or fill sections, and no material side-cast. If any new area is disturbed, any suitable topsoil will first be removed and stockpiled as described in the topsoil plan previously submitted. At the time of final reclamation, the road will be reclaimed as outlined in the previously submitted reclamation plan, unless it is determined to be necessary for post mining land use. Hydrology for the drainage controls has been calculated by Horrocks Engineers (See appendage A). Culverts will be galvanized corrigated type.

Please see enclosed map, profile, and cross-section for construction detail. (Plate 3-5)

CO-OP MINING COMPANY

Wendell Owen

WENDELL OWEN

AGREEMENT

This agreement made and entered into this 3rd day of August 1983, by and between Emery County, a body corporate and politic (County), and Co-Op Mining Company, a Utah general partnership (Co-Op),

WHEREAS, there is an existing road in Emery County known as Bear Creek Road, and

WHEREAS, Co-Op requires extensive use of said road, and

WHEREAS, due to said extensive use, said road should be relocated for the health, safety and welfare of the citizens of County as well as others who may have occasion to use said road,

NOW, THEREFORE, be it agreed as follows:

1. The parties hereto agree and acknowledge that the southern 0.65 miles of the road known as Bear Creek Road is a County road. Said County road runs from State Road 31 in a northerly direction for approximately 0.65 miles to a presently existing gate. Thereafter the road is a private road.

2. That Co-Op will relocate the Bear Creek Road according to the plans and specifications prepared by the Emery County Engineer and described on the document entitled Bear Canyon County Road Relocation dated October 12, 1982.

3. Co-Op will relocate the Road according to the plans and specifications referred to above at their expense. Co-Op will reimburse County for engineering costs incurred by County concerning the preparation of said plans and specifications and site inspections up to One Thousand ($1,000.00) Dollars.

4. Co-Op will indemnify and defend County for any damage caused, or loss incurred to or claim made by any public or private individual, firm, group, association, partnership or corporation as a result of the construction conducted to relocate Bear Creek Road. Said indemnification will continue until such time as County approves the completed roadway and accepts the construction thereof.

5. Co-Op acknowledges and accepts the easements of North Emery Water Users and Huntington City which exist in, along and across the relocate Bear Creek Road. Said easements are in existence on the ground. Co-Op's acknowledgment thereof herein recognizes and preserves said easements.

6. Co-Op agrees to encase water lines of North Emery Water Users and Huntington City in nestable corrugated pipe pursuant to plans and specifications prepared by the Emery County Engineer.

7. Co-Op agrees to allow access to other property served by the relocated Bear Creek Road. Said access shall be allowed to the owner of the property, their successor in interest or any other individual, firm, group, association, partnership or corporation who requires access due to their association with the owner or because the owner has granted permission to the individual, firm, group, association, partnership or corporation to go upon his property. Co-Op will not withhold access due to the type of activity which the then owner or his agents, employees or invitees intend or in fact conduct.

8. Co-Op will provide a completion and performance bond to Emery County upon the execution hereof in the amount of Twenty-Five Thousand ($25,000.00) Dollars which will remain in force and effect for twelve (12) months after the date said road is accepted by County as indicated in paragraph 4 above.

9. Co-Op will provide liability insurance in an amount not less than Five Hundred Thousand ($500,000.00) Dollars to be in force during the construction of said road. Said policy will name County as an insured.

10. Co-Op agrees to complete said road in a timely manner not to exceed eighteen (18) months from the date of this agreement. County may make demand upon the bonding company under the bond provided pursuant to paragraph 8 above and secure completion of the relocation in the event construction is not completed within the agreed upon eighteen (18) months.

11. It is further understood that any additional improvements of the relocated Bear Creek Road will be at the expense of all primary users.

12. The Co-Op agrees to reclaim that portion of the old Bear Creek Road according to the specifications and requirements of the Bureau of Land Manager (BLM).

13. That the Co-Op agrees to provide Emery County with the necessary easement agreements with the Utah Department of Transportation.

14. Co-Op acknowleges and agrees to comply with standard number 6.3.8 "Protection Zone" of the Utah State Health Drinking Water Standards as it applies to supplies of drinking water in Bear Canyon.

15. County agrees to inspect the relocated Bear Creek Road within ten (10) days after notification by Co-Op of the completion thereof. County must within five (5) working days of said inspection accept the road or notify Co-Op of any deficiencies which must be then corrected by Co-Op within the time period outlined in paragraph 10 above. Should County fail to notify Co-Op of any deficiencies within five (5) working days, the road is deemed accepted by County and the twelve (12) month period indicated in pargaraph 8 above begins to run from the sixth (6th) day after inspection.

IN WITNESS WHEREOF, this agreement is executed the day and year above first written, at Castle Dale, Utah, pursuant to a resolution of the Emery County Board of Commissioners at a regularly scheduled meeting of the Board.

EMERY COUNTY, a body politic and corporate,

ATTEST

County Clerk

By
Chairman of the Emery County Board of Commissioners

IN WITNESS WHEREOF this agreement is executed at Huntington, Utah. DATED this 3rd day of August, 1983.

CO-OP MINING COMPANY, a Utah general partnership

By B W Stoddard
a General Partner

Appendix 5-G

Tank Seam Access Road

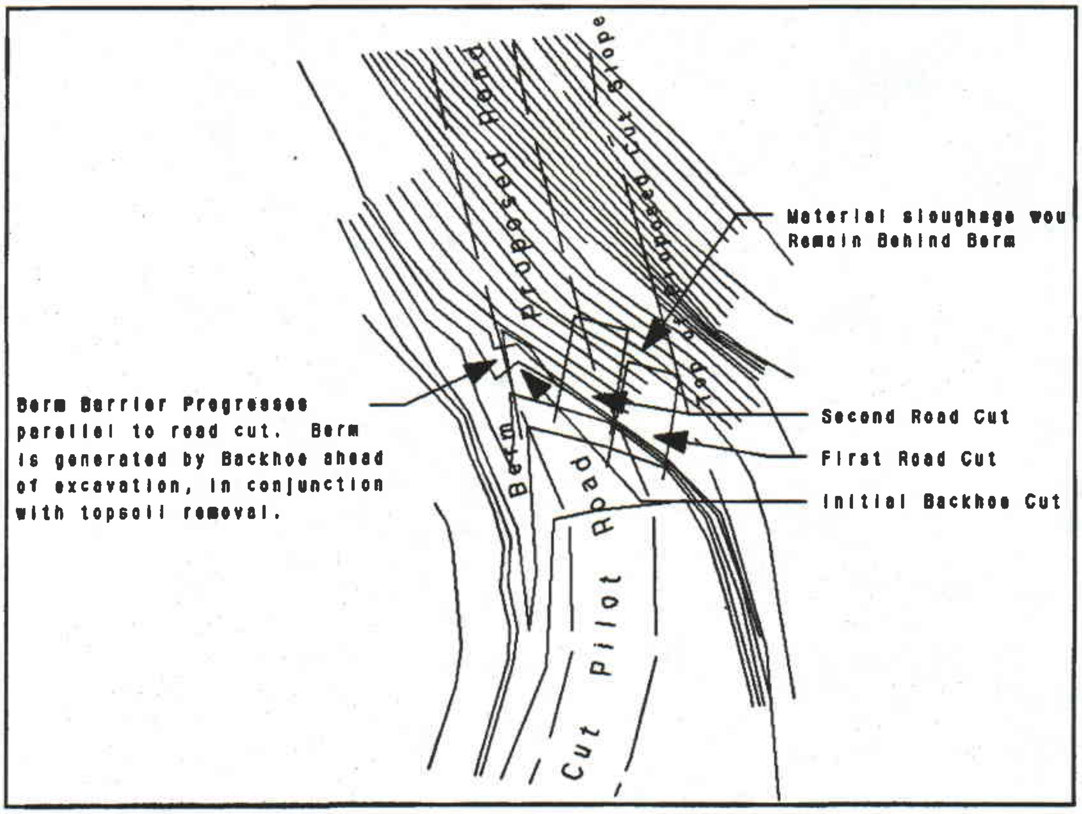
This appendix discusses the construction of the Tank Seam Access Road and Tank Seam Portal Pad. In order to clearly complete proper out and fill calculations for the road, the road has been divided into stations spaced at approx. 100 ft o.c. as shown in Figure 3H-4. A cross section of each station follows Figure 5G-4. Stations 0+00 and 1+00 lie on the Upper storage Pad, Station –1+00 to the North on the existing road and Station 2+00 to the south on the lowest end of the Tank Seam Road. The station numbers increase as you proceed up the road toward the pad.

Construction will begin on the Upper Storage Pad. Construction will consist of removing topsoil for storage, then removing material from the road cut and compacting in onto the Upper Storage Pad. Topsoil removal and storage is discussed in section R645-301-240. Table 5G-1 shows the cut and fill volume calculations. Volume in the table represent total cut and fill material will be placed in the area of stations 8+00 and 9+00.

During the initial road cut, care will be taken to prevent disturbed material form migrating downslope in the following manner. The initial topsoil removal will be made using a backhoe. Trees and/or shrubs immediately ahead of the cut will be removed by pulling them back into the previous cut. Using the backhoe, a berm will be created on the downhill side of the cut, as shown in Figure 5G-1. When the berm is in place, the road cuts will be started as shown in Figure 5G-1 using a backhoe and/or front-end loader. The road cuts will be made into the slope towards the cut face rather than parallel to the slope, which will result in any rocks or sloughage dislodged by the equipment bucket during the road cutting to be contained within the berm. In the event blasting is required, which is described in Appendix 5-E, the blasts will be designed to drop the material in to cut area behind the berm. This will prevent material generated by the blast form migrating downslope into the undisturbed area. This procedure will de used to cut a pilot road until the first large fill area. This procedure will be used to cut a pilot road until the first large fill area (Stations 8+00 and 9+00) is reached.

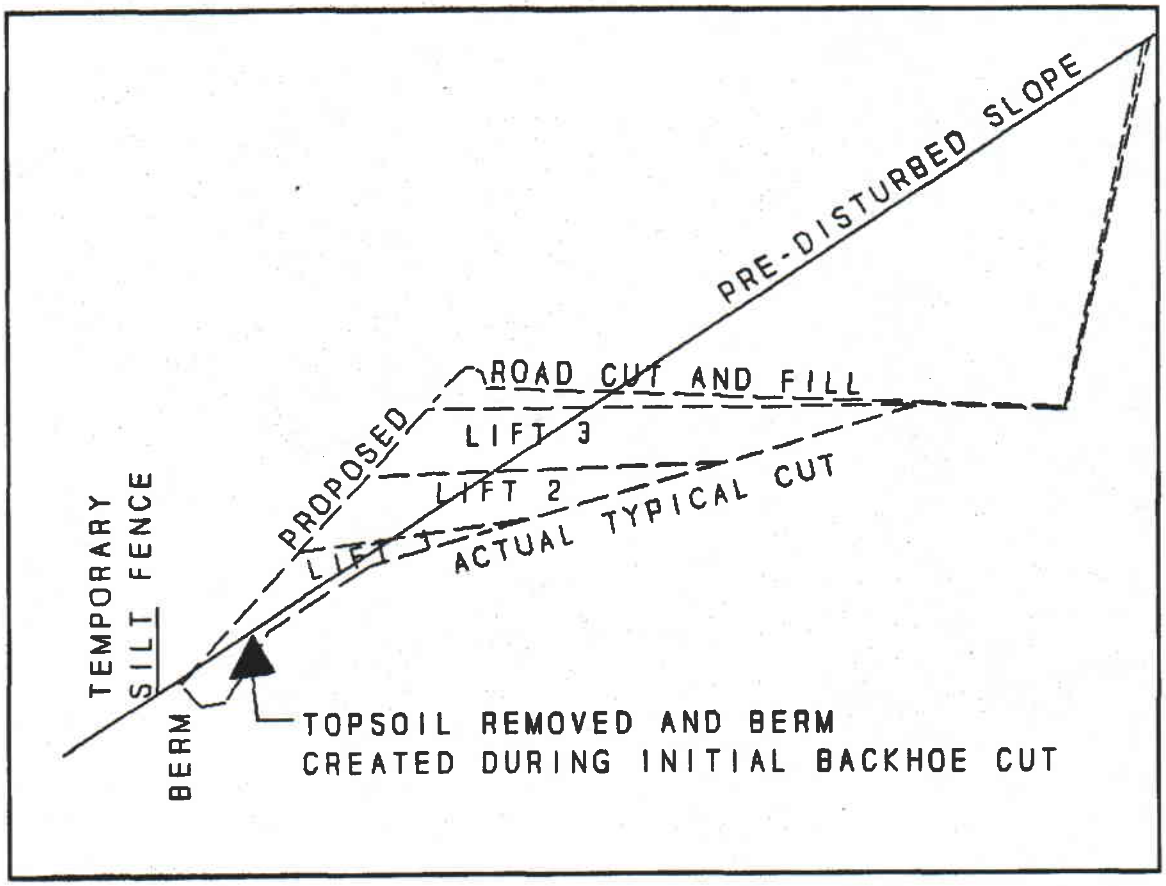
When small fill areas are reached (e.g. Station 6+00), a temporary silt fence will be installed at the base of the proposed fill for runoff control, and the same cutting procedure will be used to create an initial berm inside the silt fence with a backhoe after topsoil removal. The area inside the berm will then be prepared to allow the placement of the fill, as shown in Figure 5G-2. Fill material in these areas will be restricted to no more than 15' downslope form the road, allowing a backhoe to easily reach the material during reclamation. Rock fragments larger than 18 inches which are disturbed will be embedded into the surface of the fill as described in the slope stability analysis on page 5G-48. These areas are included in the disturbed area, and are designated as BTCA areas (See Plate 7-1E and Appendix 7-K). As soon as the fill material is in place, erosion control matting will be placed on the disturbed slopes as described on Appendix 7-K. The temporary silt fence at the base of the fill material will remain in place for runoff treatment until installation of the erosion control matting is complete.

Figure 5-G-1. Typical Road Excavation Sequence



NOTE: Road cut sequence may vary. All cuts will be made behind the berm generated by the initial backhoe cut.

Figure 5G-2. Cut and Fill Diagram for Small Areas



NOTE: This diagram represents a typical sequence for the construction of small fills alongside the cut road. Configurations shown are typical and the actual profile may vary.

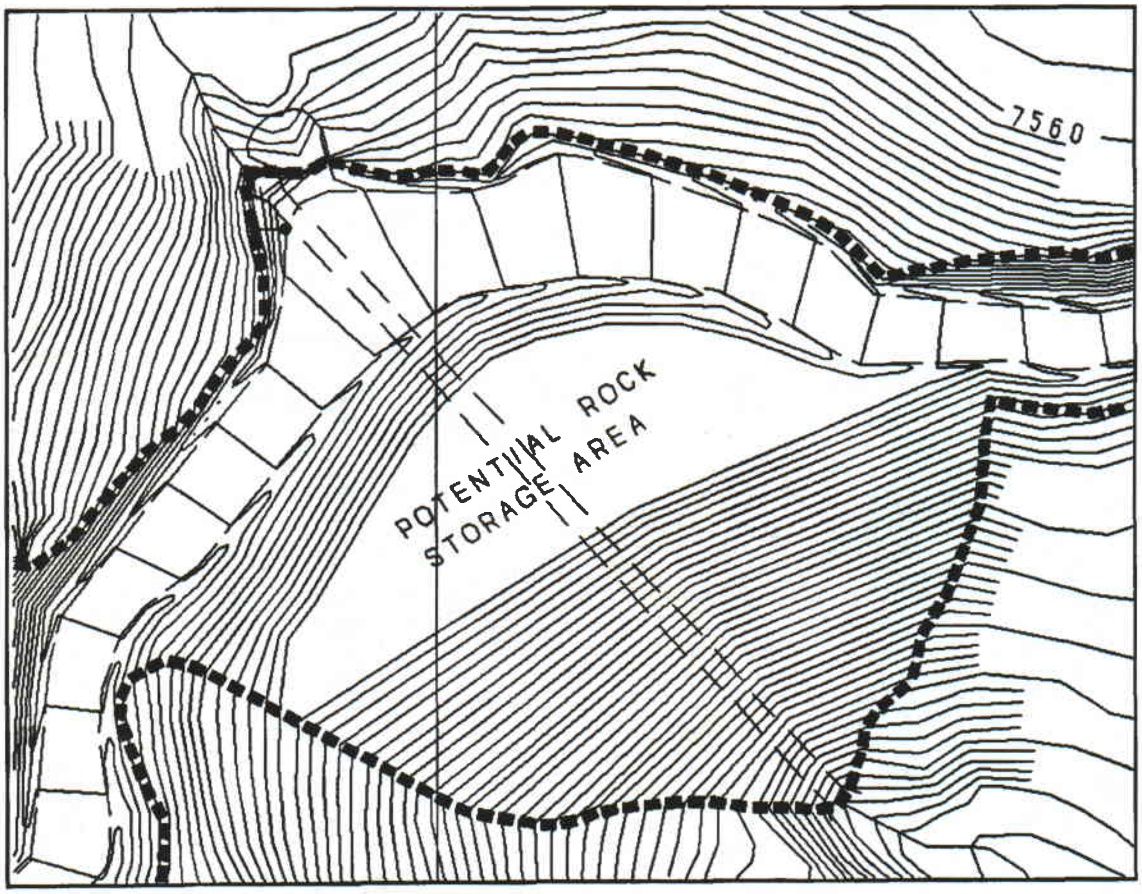
When the large fill area is reached, temporary silt fences will be placed below the fill area prior to equipment entering the proposed disturbed area. The silt fences will be placed so as to treat runoff from all disturbed area not contained by a berm during construction. Topsoil recovery will then begin in the fill area. During topsoil removal a ramp will be constructed which will weave back and forth along the face of the fill area to the base.

As the base of the fill area is approached, care similar to the procedures described previously for cut and fill will be taken to prevent rocks and other material form being pushed onto or below the silt fences. The following sequence will then be used starting at the base of the fill area.

The base of the fill area will be prepared according to the recommendations on Page 5G-48 by removing all vegetation and rock fragments larger than 18 inches which are not embedded into the natural ground and/or stable, and any cobble or boulder sized rocks which are positioned so as to interfere with the compaction activities. Placement of the fill material will then begin and the material will be compacted in lifts not exceeding 18 inches. The initial cut to reach the base of the fill area will act as a series of terraces with which the fill material can be keyed into the natural soils, as recommended on page 5G-48. As the fill progresses up the slope, removal of rock fragments and vegetation will continue on the slope above the fill. Rock fragments less than 18 inches will be incorporated into the fill as they are removed. Rock fragments larger than 18 inches will be incorporated into the surface of the fill and will be embedded into the fill material to aid the surface, remaining rocks will be temporarily placed on the Upper Storage Pad until construction of the road is complete, and then will be placed as described below according to the discussion on page 5G-48.

The cut and fill volume in Table 5G-1 represent the maximum amount of fill which will be encountered in the road excavation. The possible presence of excessive rock fragments larger than 18 inches may generate a reduced amount of fill and rocks which may need to be stored until reclamation. The extreme width of the road at stations 8+00 and 9+00 account for the maximum amount of fill material, which may be encountered. Excessive rock fragments could decrease the available fill. If this scenario is encountered, an alternate configuration, involving terracing of the fill, would provide a stable surface for which to store the additional rocks encountered during construction. If additional storage area is required, the necessary amount of area will be obtained by redesignating part of the Upper Storage Pad for storage of these rocks. These storage areas will be level or near level, so as to meet the requirements of R645-301-535. Rock fragments will be placed in single lifts as described on page 5G-48, in order to guarantee the safety factor requirements of R645-301-535.

Figure 5G-3. Alternate Fill Area Configuration.



Scale: 1" = 40'

Note: This figure represents an alternate fill configuration to account for the potential lack of fill material due to excessive large rocks (>18" m.d.). The elevation of the terrace may be lower than shown, or the terrace area may consist of more than one terrace, depending on the amount of fill available and amount of rocks encountered. As excavation progress, the amount of fill will be monitored. Approximately 3900 sq. ft. of terrace area is available for potential rock storage area (0.09 acres).

As the fill material is placed and compacted, the surface of the slope will be scarified and ripped horizontally to create a rough surface and water-holding pockets for interim vegetation, along with the embedded rocks. Erosion control matting will be installed in junction with the filling and compacting to prevent erosion form occur9ng on the fresh slopes. The matting will provide erosion and runoff control, and will aid in the establishment of interim vegetation. The matting will be maintained on the slopes as the primary BTCA treatment (Appendix 7-K), and will be maintained until an adequate demonstration that interim vegetation qualifies the slopes for designation as SAE (Small Area Exemption) areas. Division approval will be required before this designation can be made. Temporary silt fences will be removed once the installation of the matting is complete.

Culverts will also be installed on the fill slope as construction progresses up the slope, in order to avoid disturbance of the surface after construction. Culvert outlets will be protected as described in R645-301-742.300.

The cut and fill methodologies in the Appendix will be used throughout the construction of the road and the portal pad. As construction of the pilot road progresses, temporary ditches which meet the designs specifications of the permanent ditches will be maintained along the pilot read, with silt fences placed just above the culvert inlets treating any runoff. Approximate silt fences locations are shown on Plates 7-1C and 7-1E. Upon completion of construction, final as-built contours will be submitted to the Division.

Final crowning of the road and installation of permanent ditches will be completed following initial road and pad contouring. The approximate road and pad contours are shown on Plates 5-2.

A slope stability analysis of the cut slopes and fill areas, as well as some discussion on the construction methodology, is on page 5G-44 following the cross sections.

Upon completion of regarding activities, interim stabilization of the cut slopes will be accomplished through hydroseeding as described in R645-301-331. Cut slopes will be seeded using the seed mix an mulch described in Tables 3-3 and 3-4. Downslopes will be seeded by hand prior to the placement of erosion control matting using the permanent seed mix shown in Table 3-3. This seed mix will be used in order to establish shrubs as well as grasses to aid in interim stability.

The final as-built cut and fill material volumes are shown in Table 5G-2 on pg. 5G-71.

During reclamation 1,000 yds³ of material was hauled from TS-15 as described on page 5J-13. Because of this 1,000 yds³ of material will be left in TS-8 for use in reclamation in that area.

Figure 5G-4. Tank Seam Road/Pad Stations Map

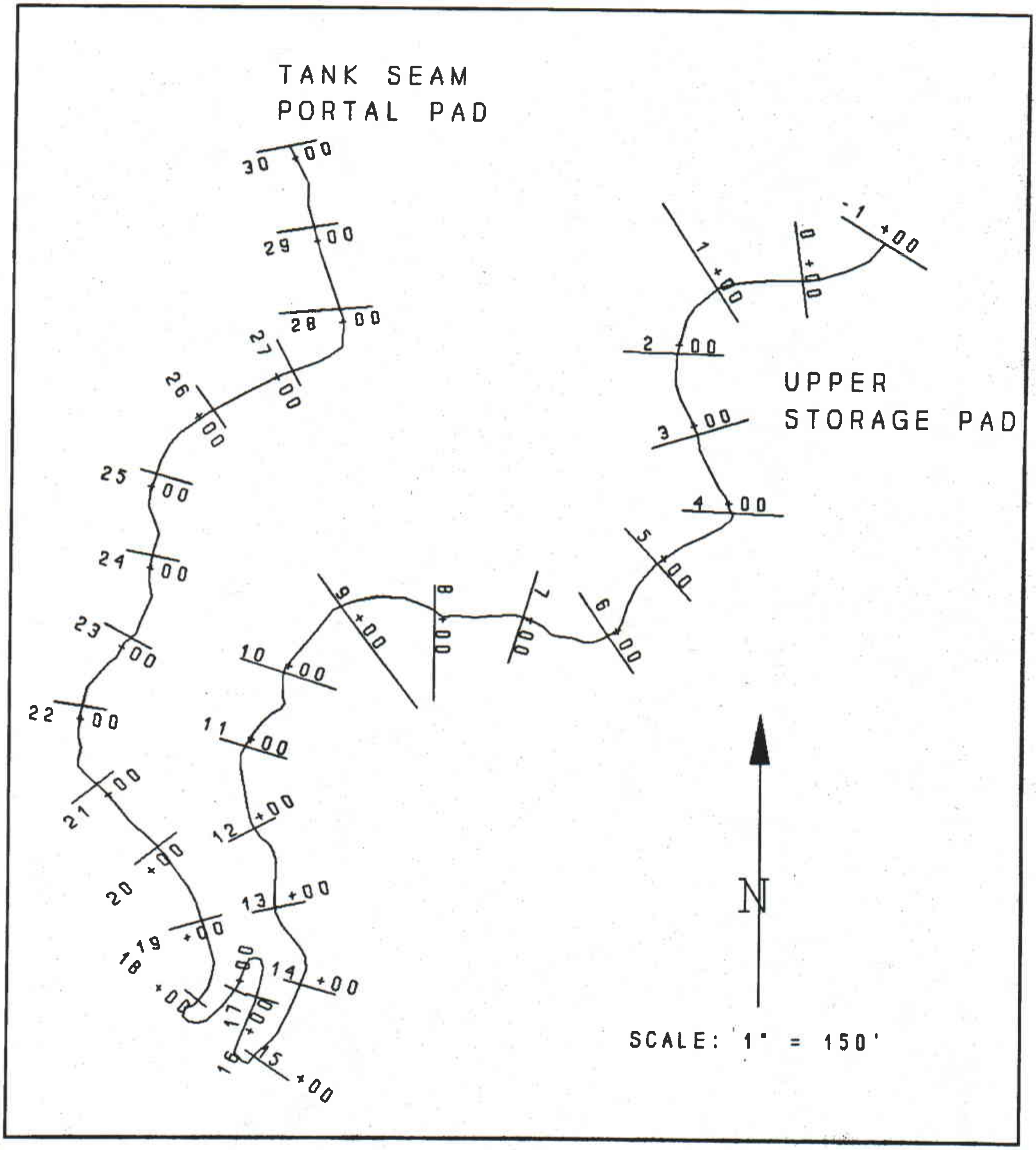
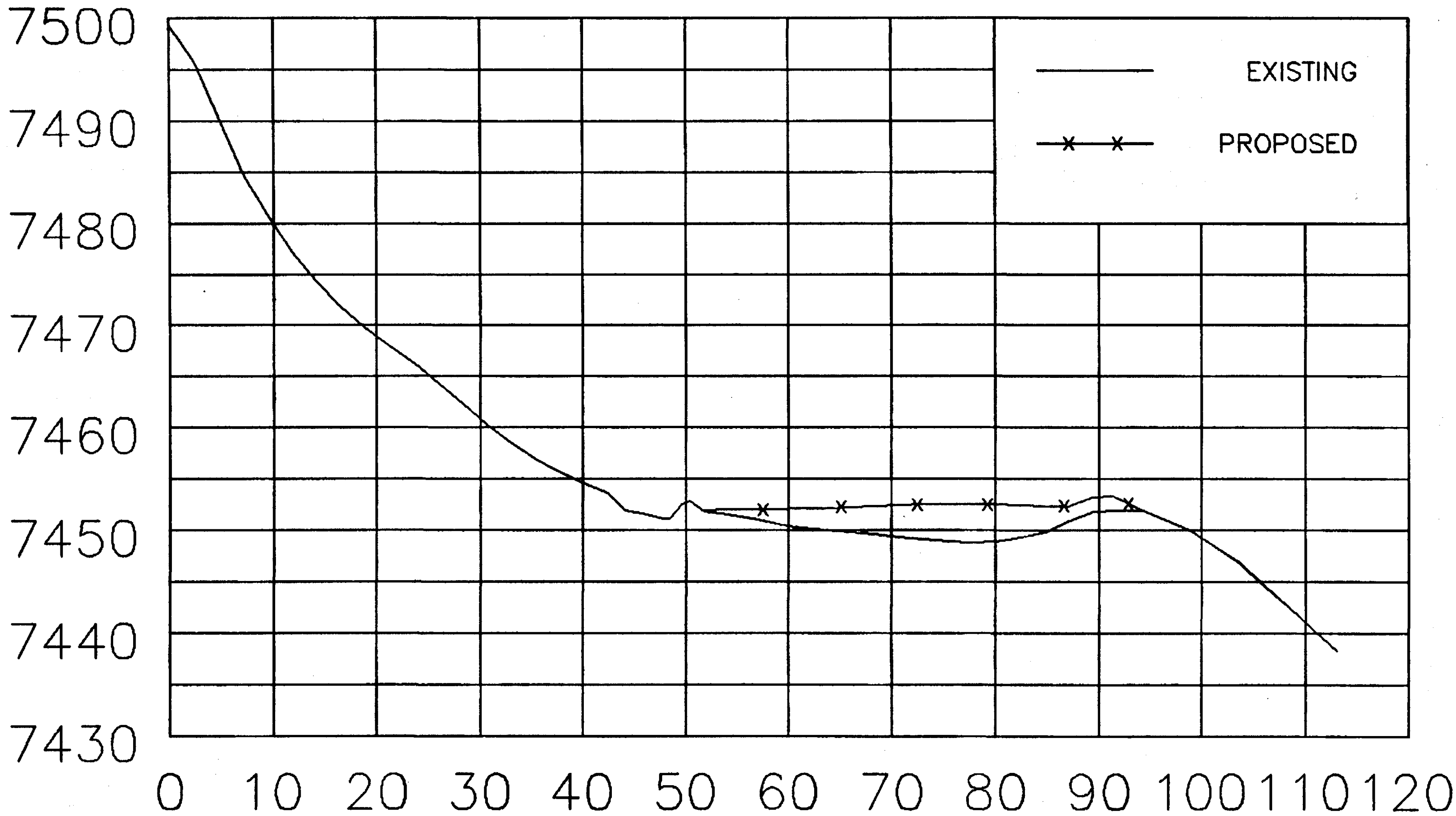


Table 5G-1. Tank Seam Road Cut & Fill Summary

Station	Fill (-) Volume (cu yd)	Cut (+) Volume* (cu yd)	Volume Cumul. (cu yd)	Station	Fill (-) Volume* (cu yd)	Cut (+) Volume* (cu yd)	Volume Cumul. (cu yd)
-1+00			0	15+00			-12,764
0+00	456	0	-456	16+00	300	69	-12,995
1+00	2,084	0	-2,084	17+00	140	236	-12,899
2+00	2,419	0	-4,959	18+00	175	299	-12,775
3+00	631	367	-5,223	19+00	0	550	-12,225
4+00	68	802	-4,489	20+00	0	871	-11,354
5+00	53	669	-3,873	21+00	0	796	-10,558
6+00	139	386	-3,624	22+00	0	819	-9,739
7+00	68	480	-3,214	23+00	0	642	-9,097
8+00	3,025	122	-6,117	24+00	0	713	-8,384
9+00	7,145	0	-13,262	25+00	822	83	-9,123
10+00	498	526	-13,234	26+00	269	536	-8,856
11+00	779	258	-13,755	27+00	0	1,265	-7,591
12+00	0	676	-13,079	28+00	0	4,528	-3,063
13+00	0	741	-12,338	29+00	0	1,994	-1,069
14+00	150	266	-12,222	30+00	0	1,110	41
15+00	1,089	547	-12,764		20,310	20,351	

*Cut and fill volumes measured using Quicksurf Version 4.0 3-D molding software package, copyright 1991, Schreiber Instruments, Inc., based on premining contours from 1991 aerial survey and proposed contours shown on Plate 5-2E.

STATION 0+00



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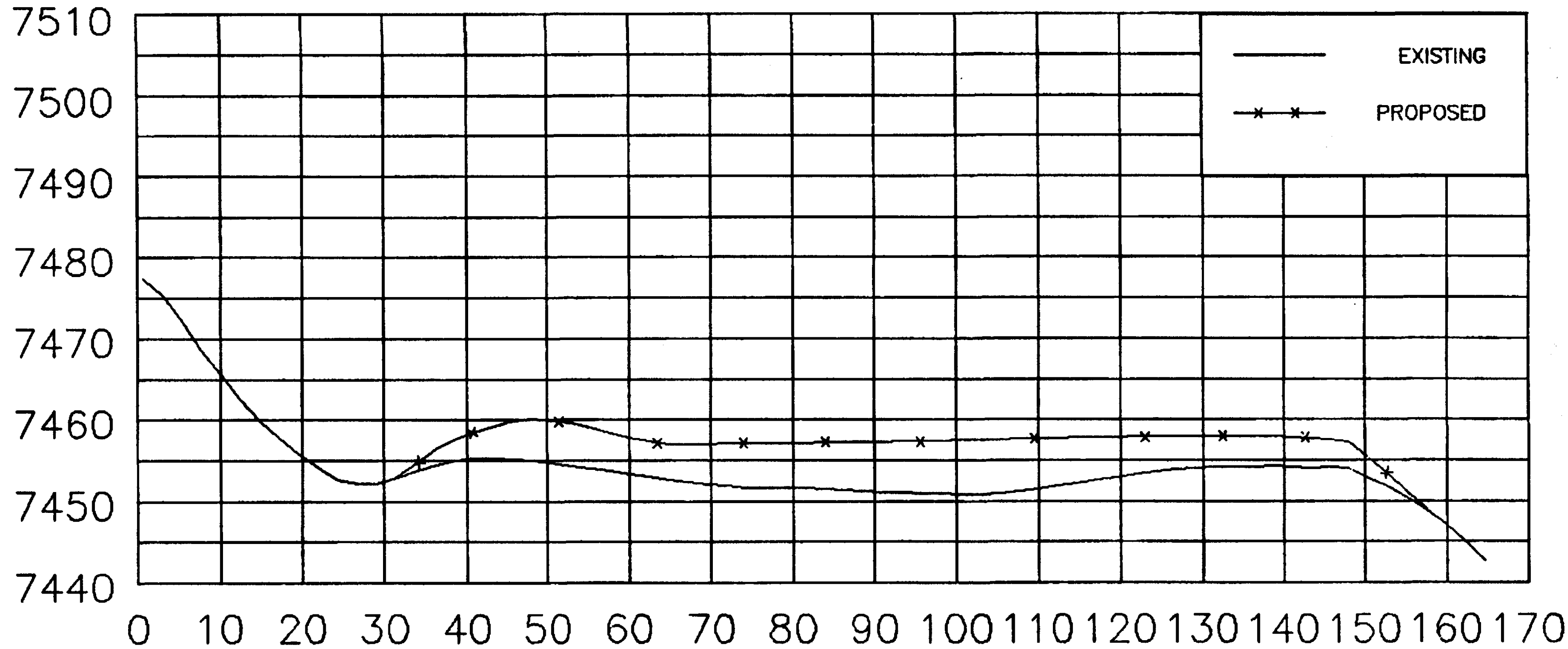

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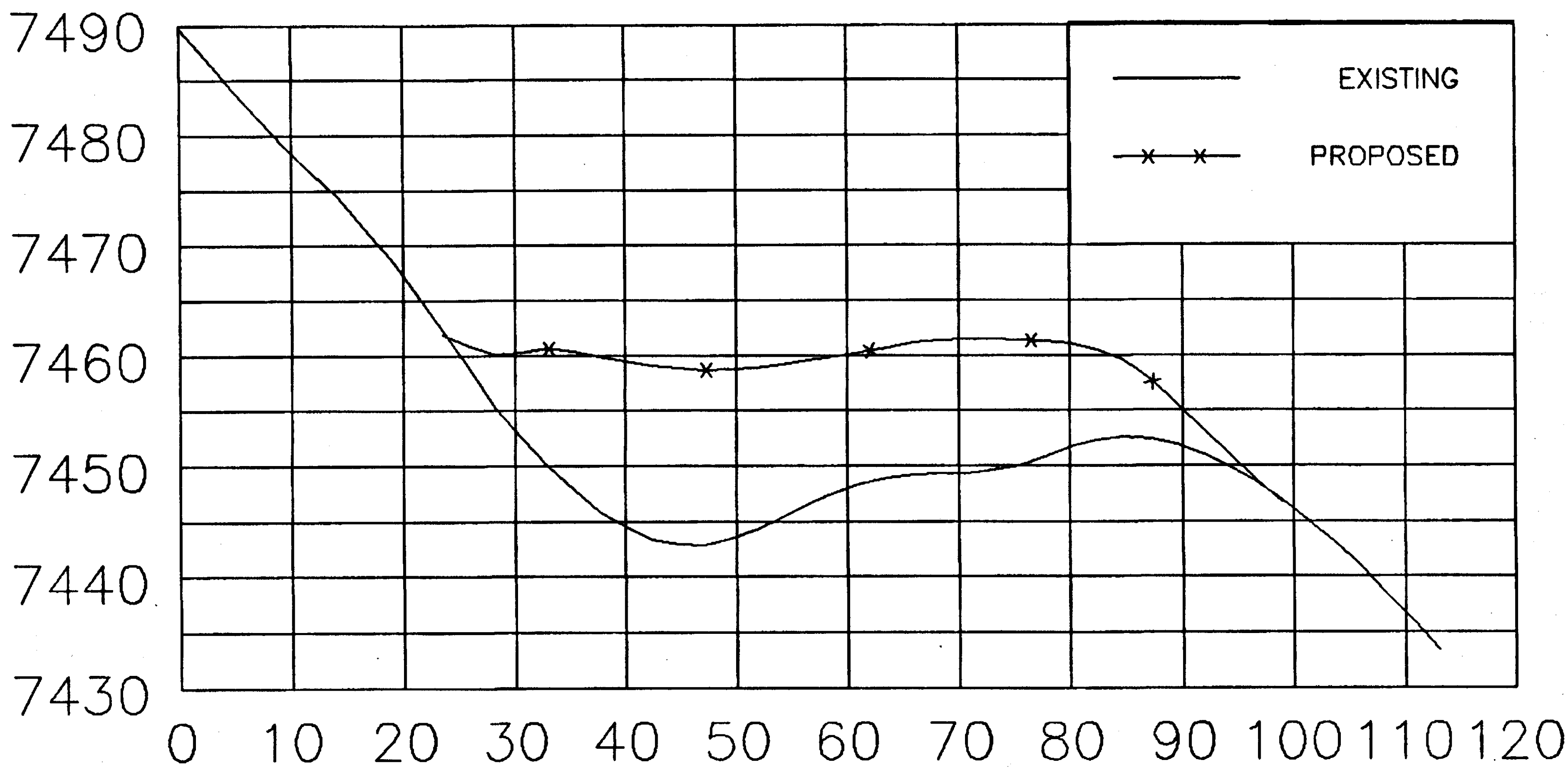
STATION 2+00



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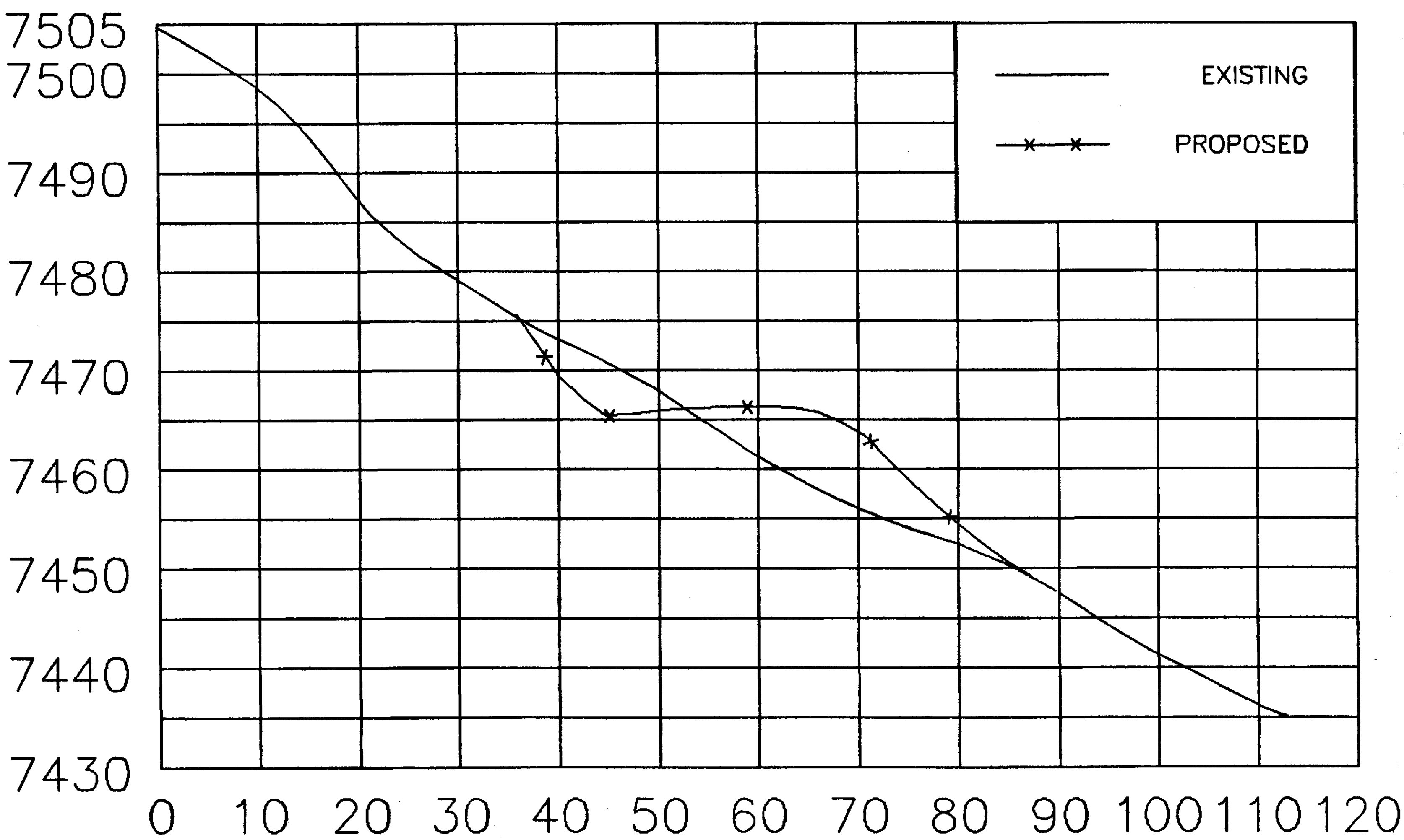
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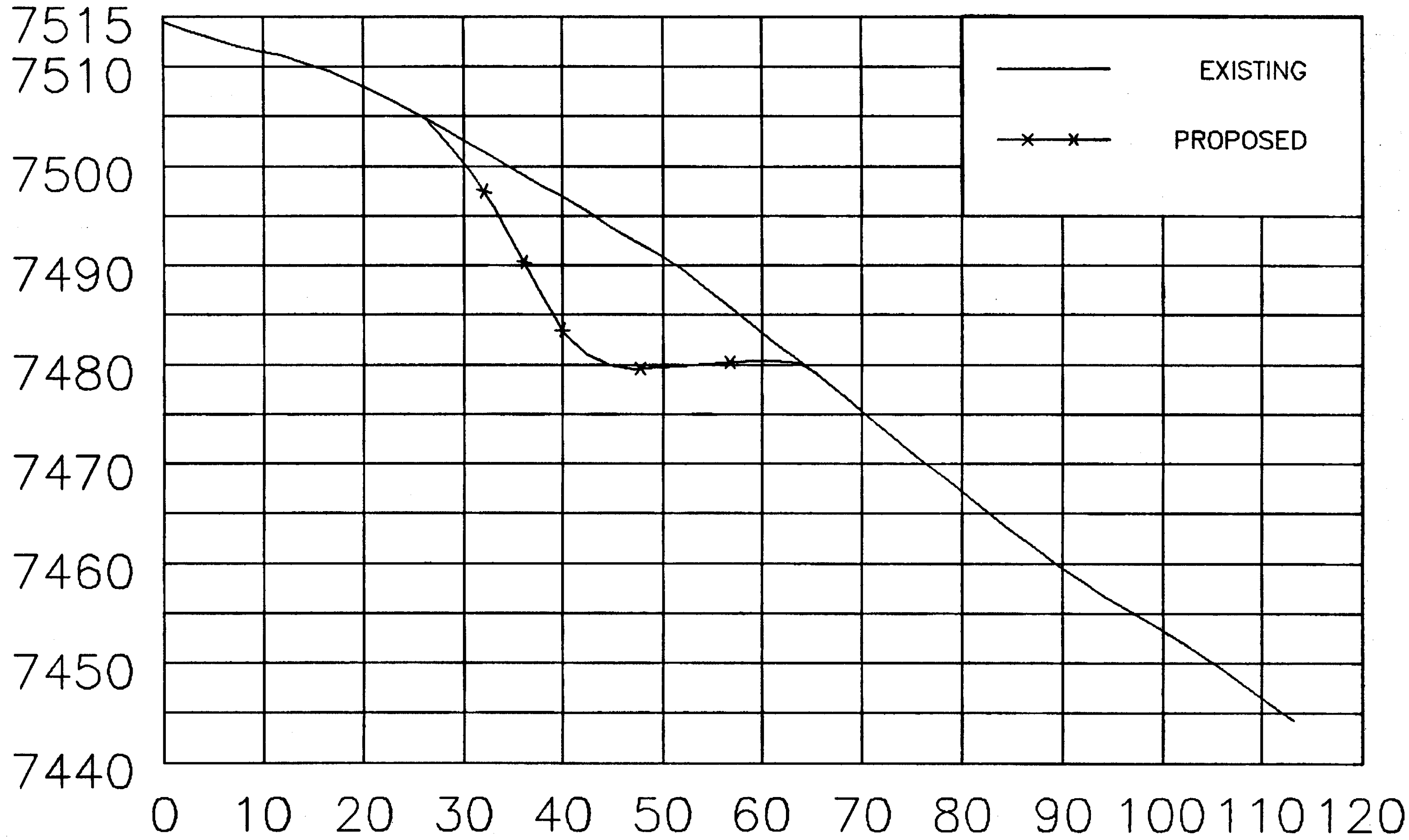
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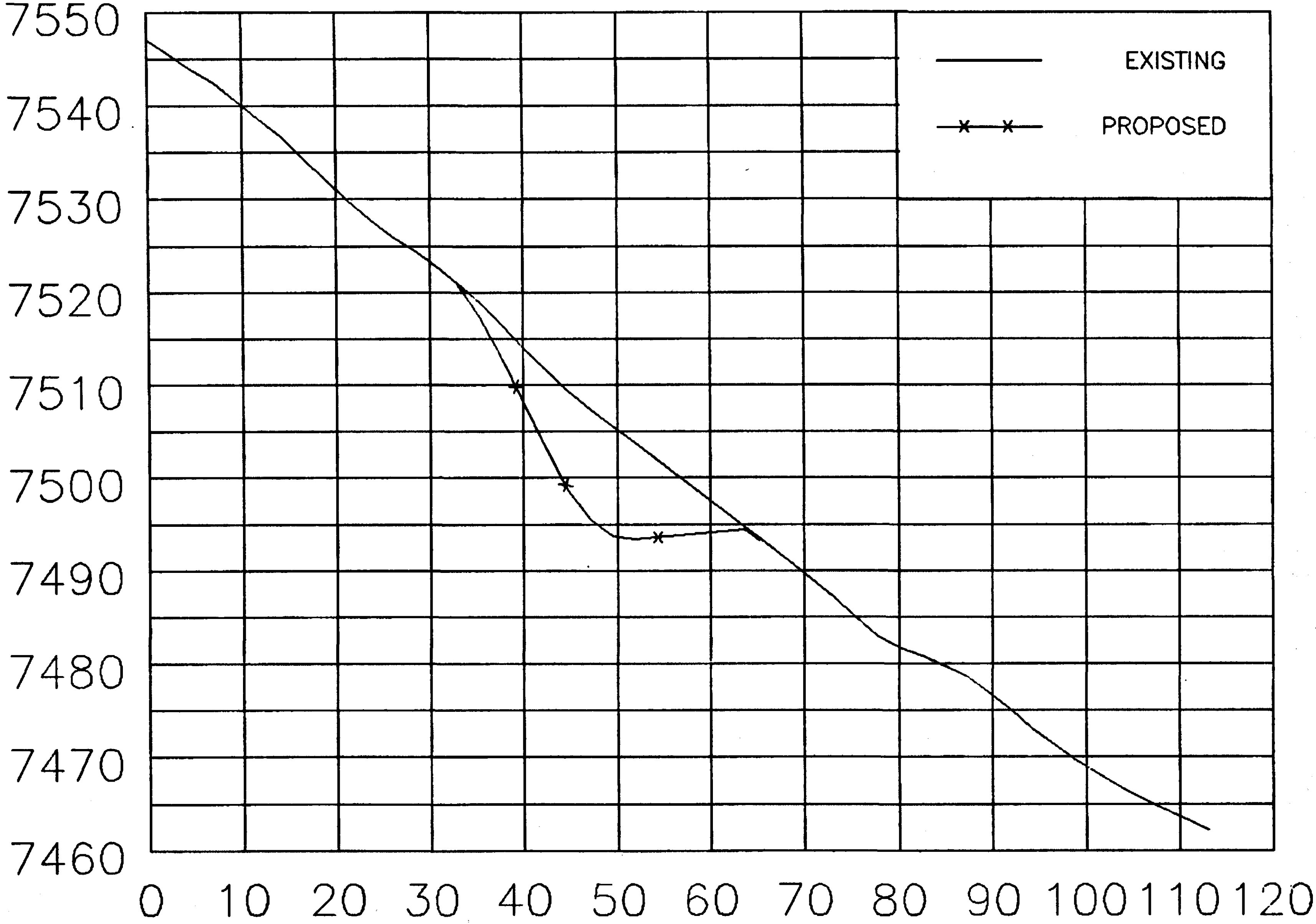
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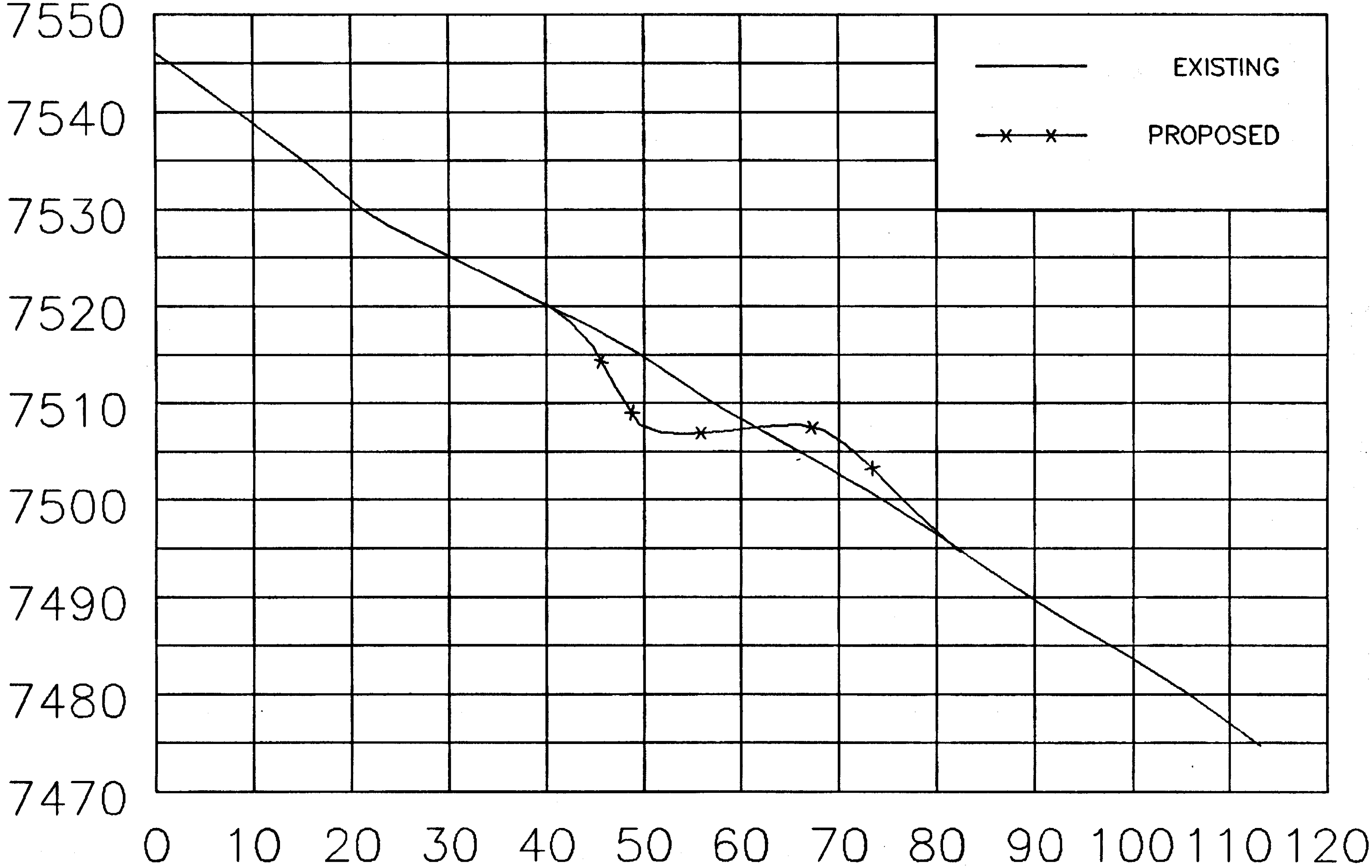
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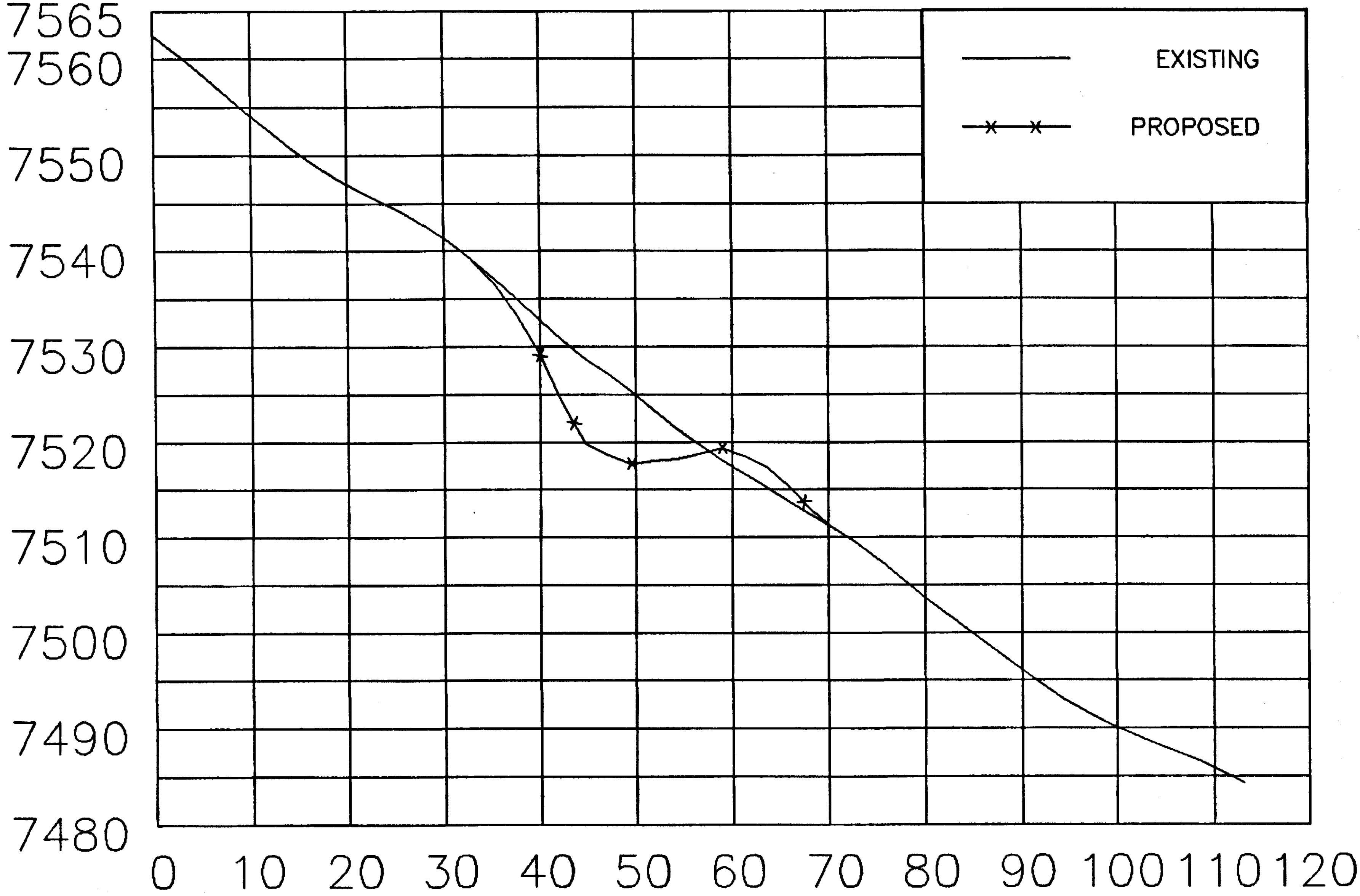

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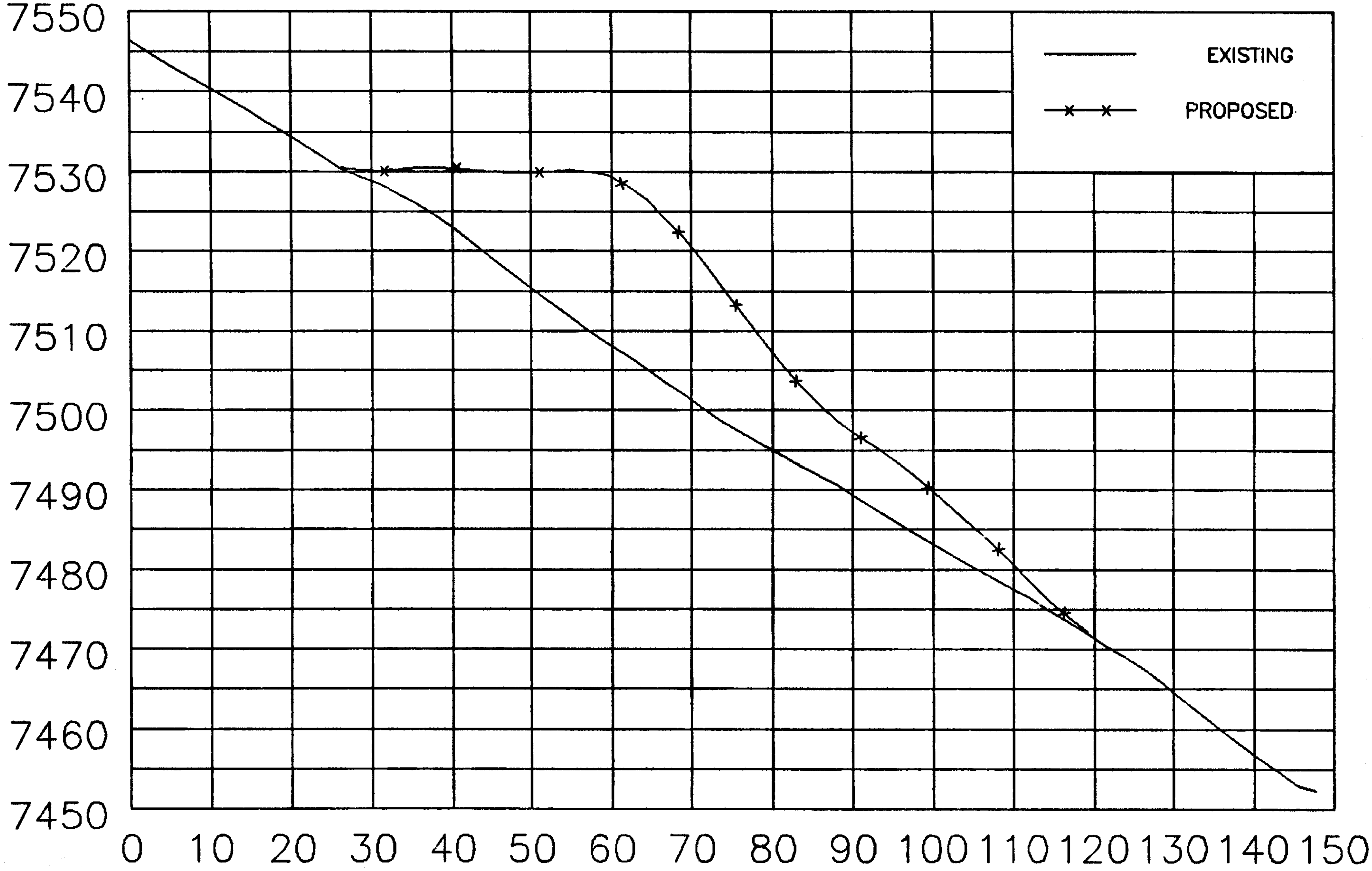
STATION 8+00



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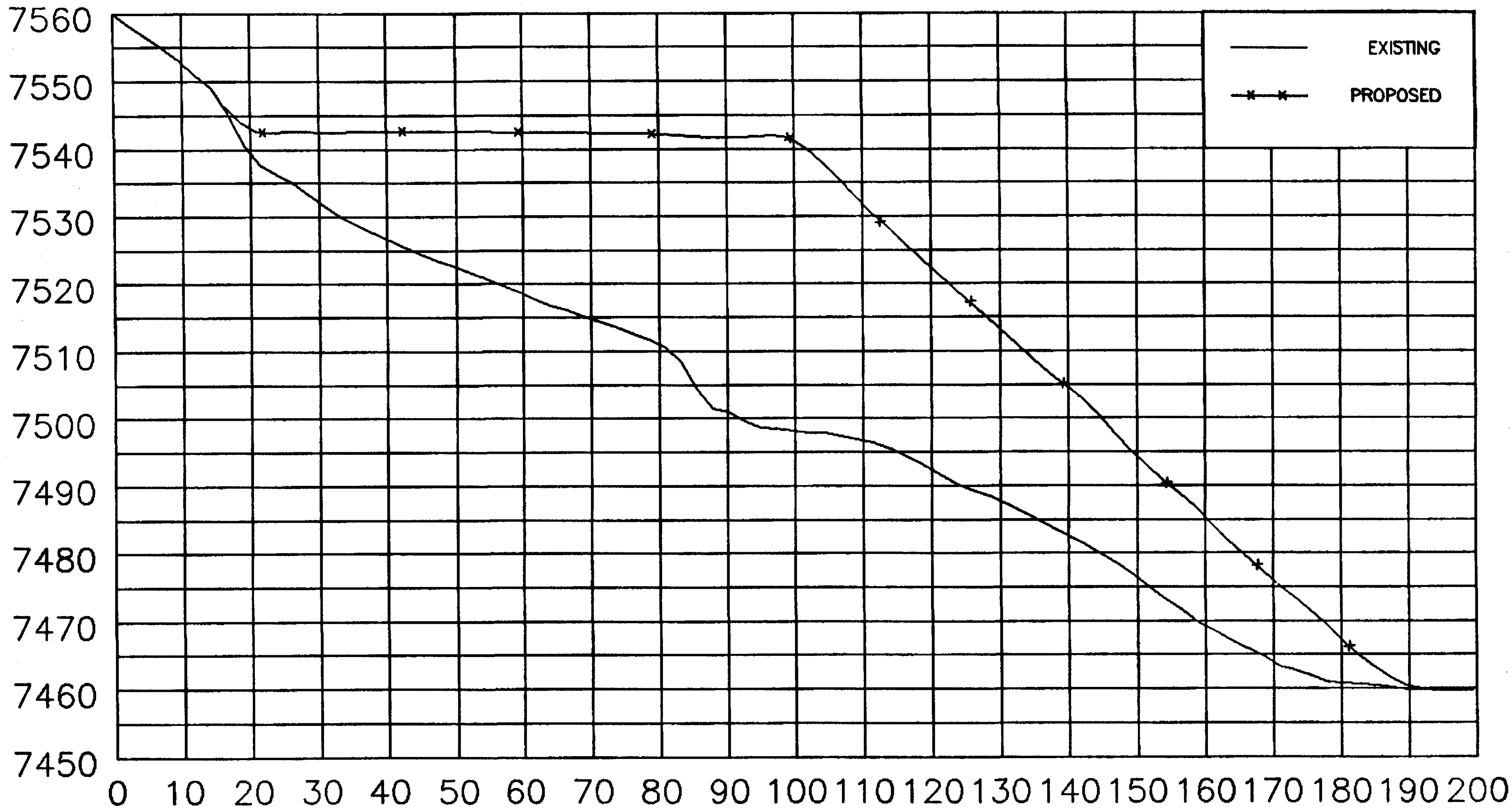

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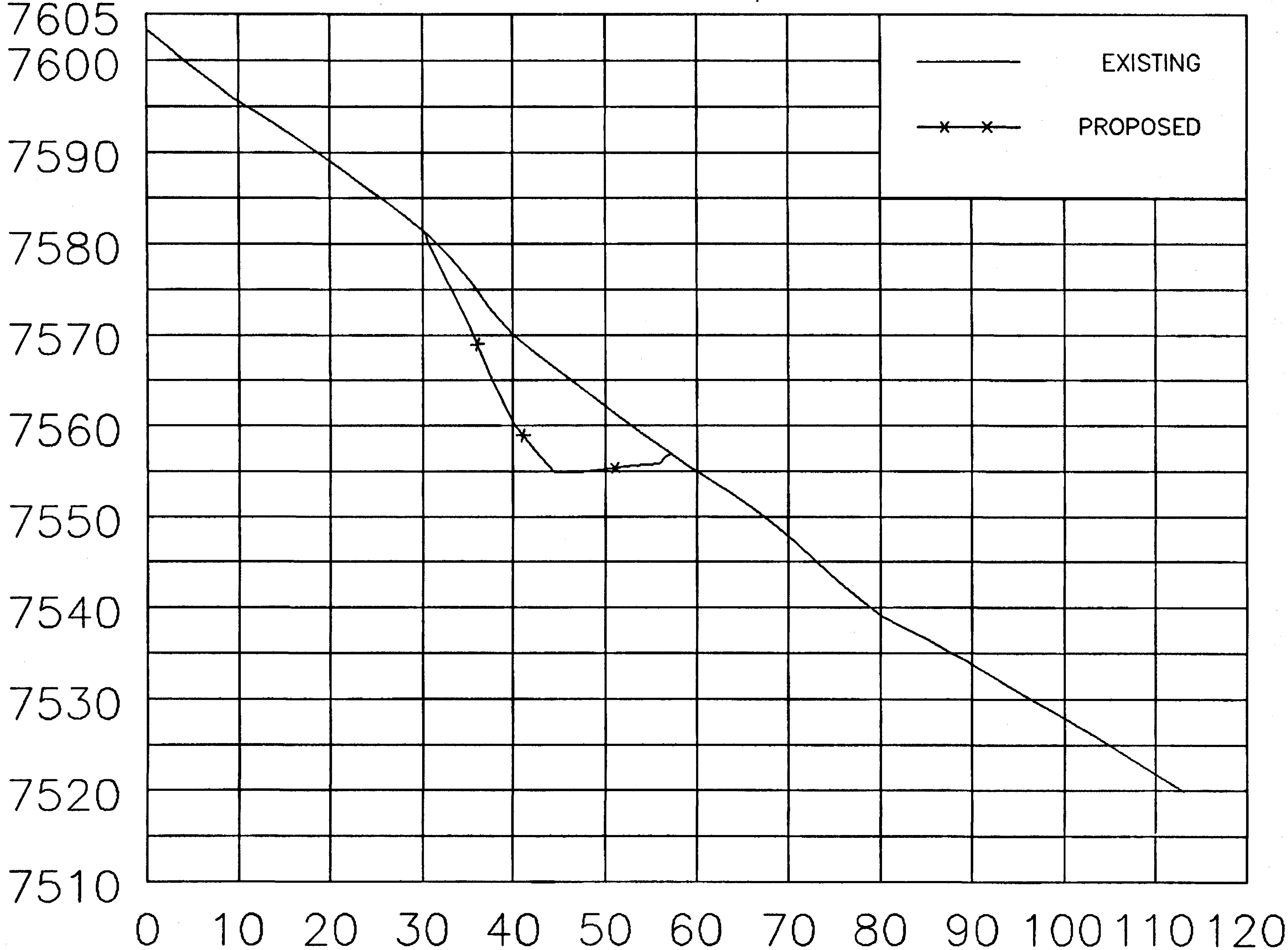
STATION 10+00



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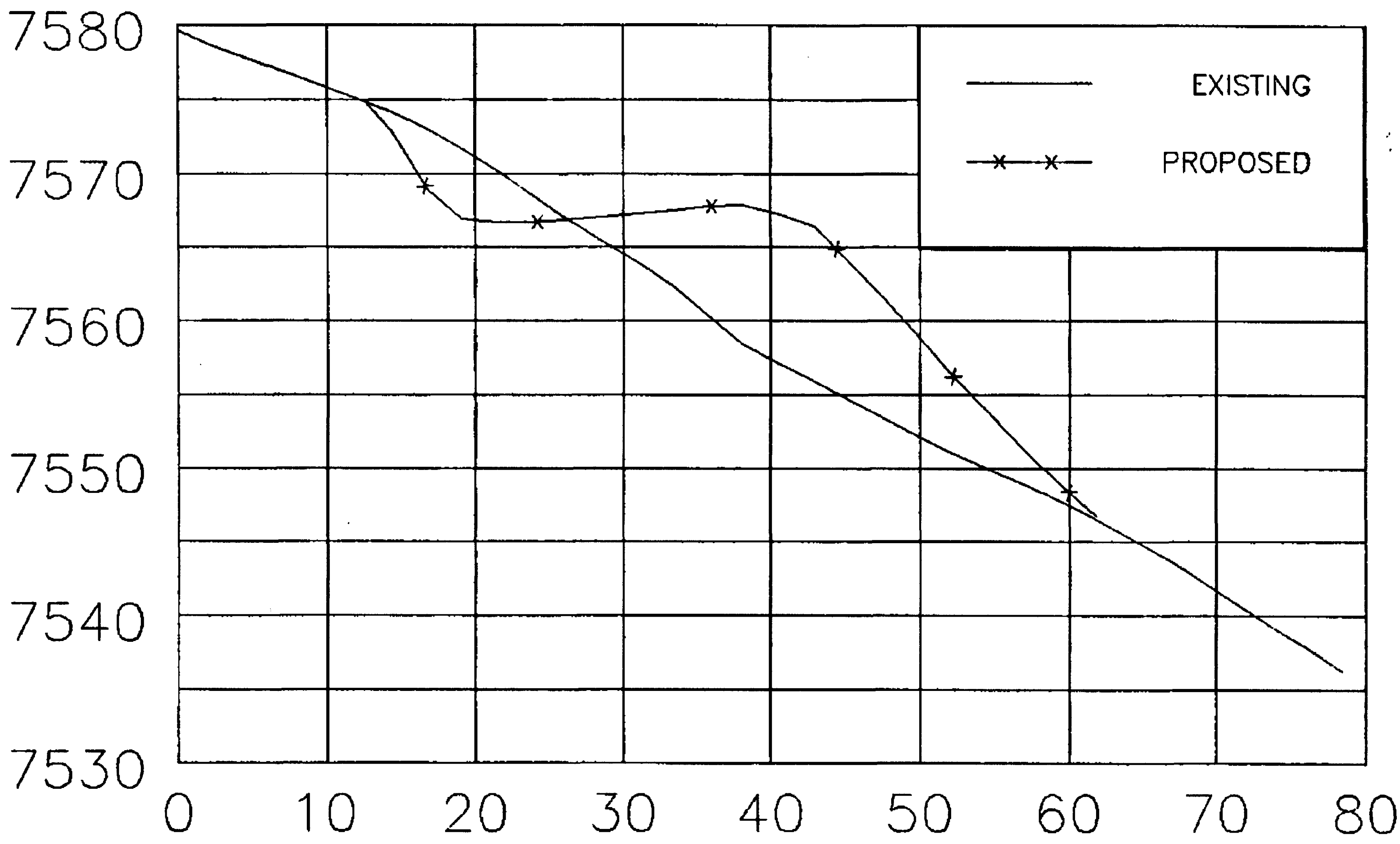

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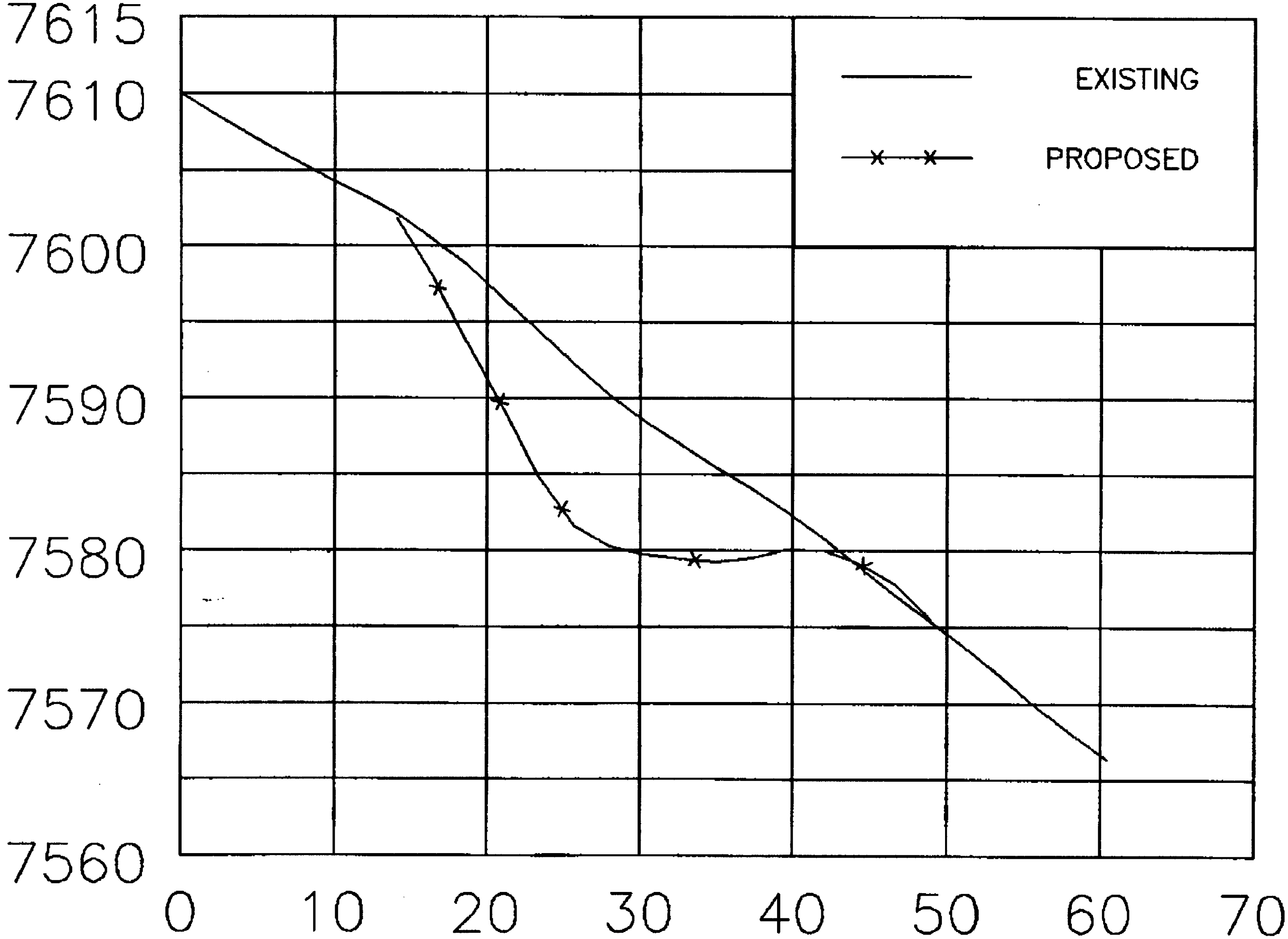

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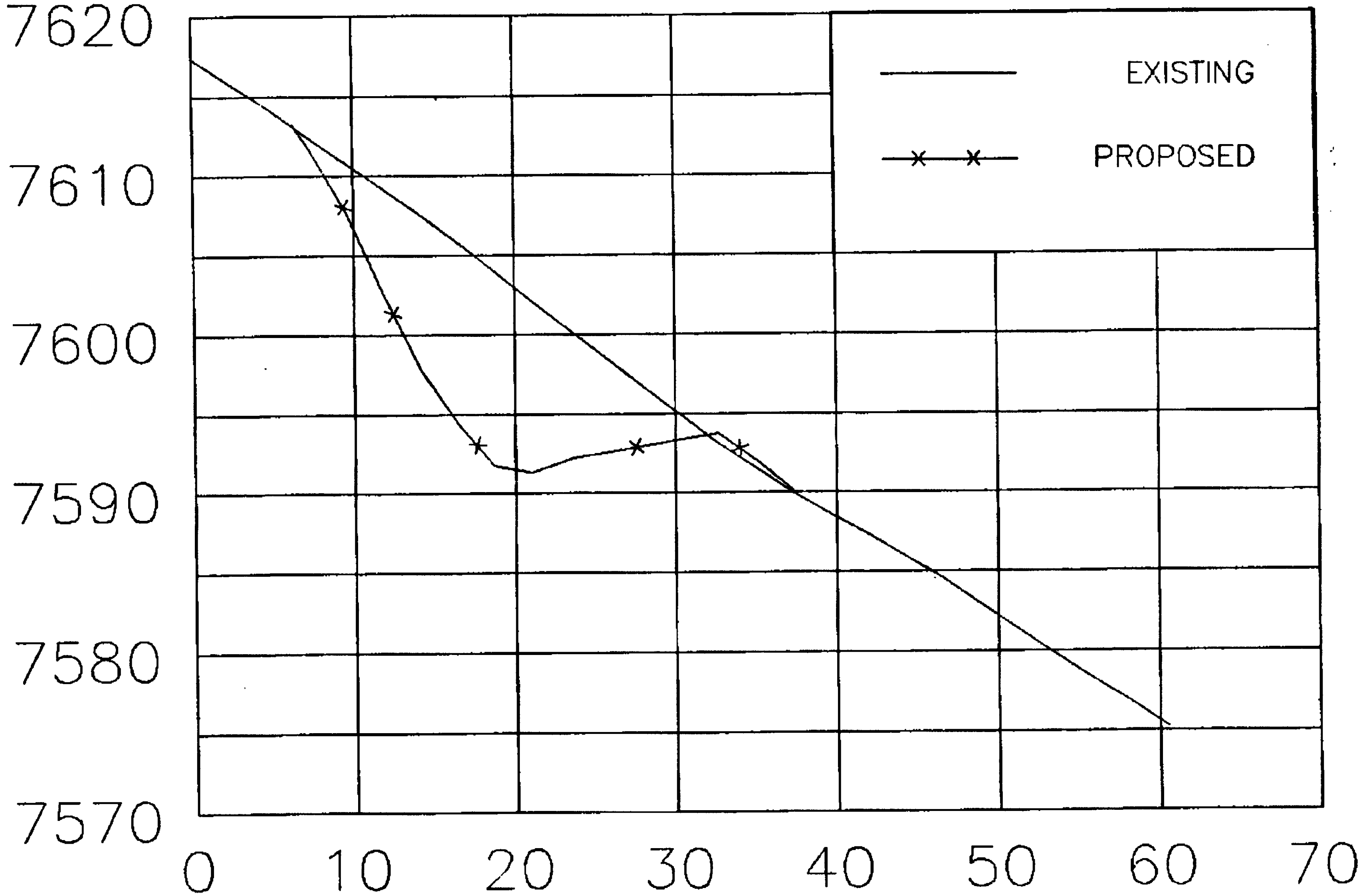
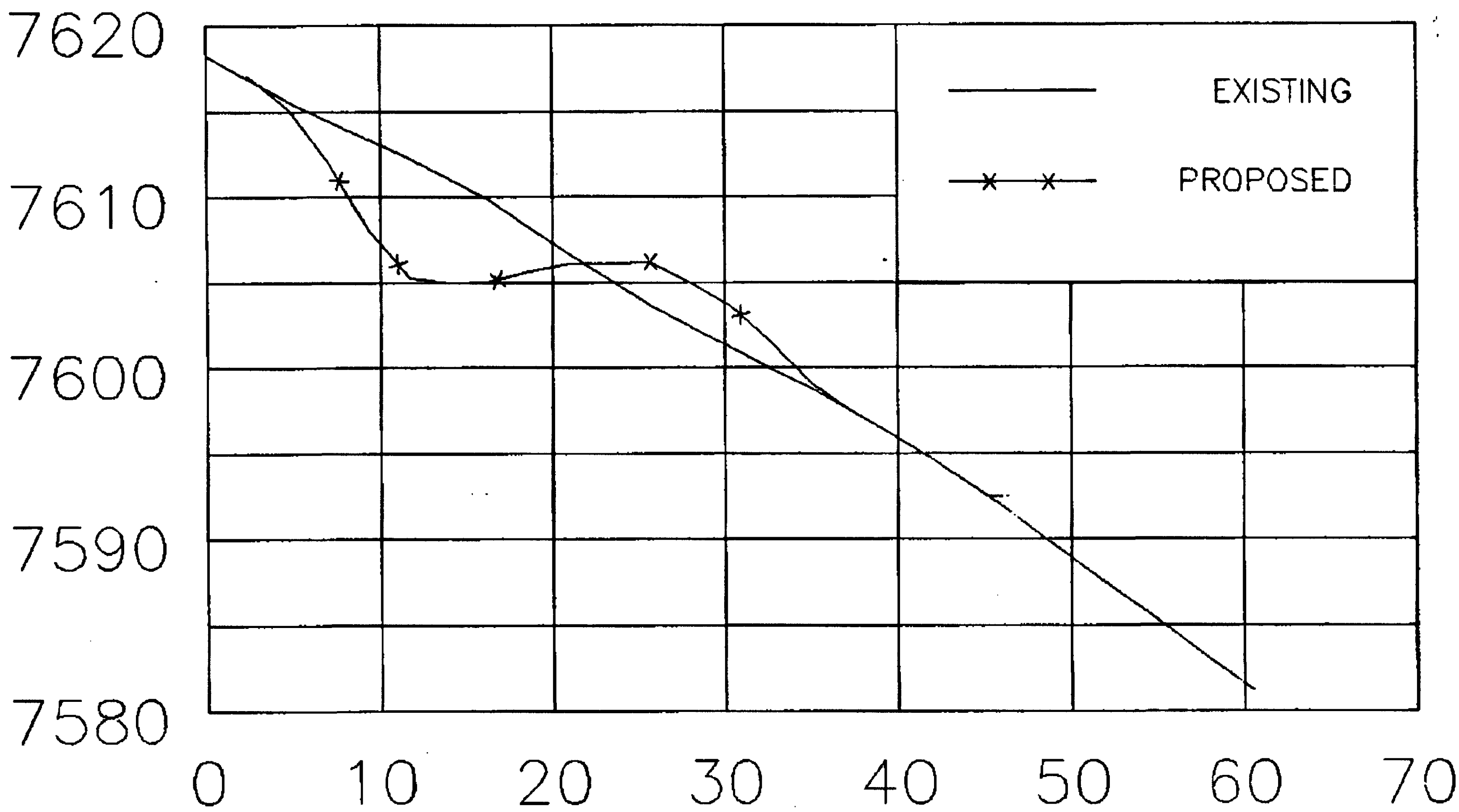


STATION 14+00



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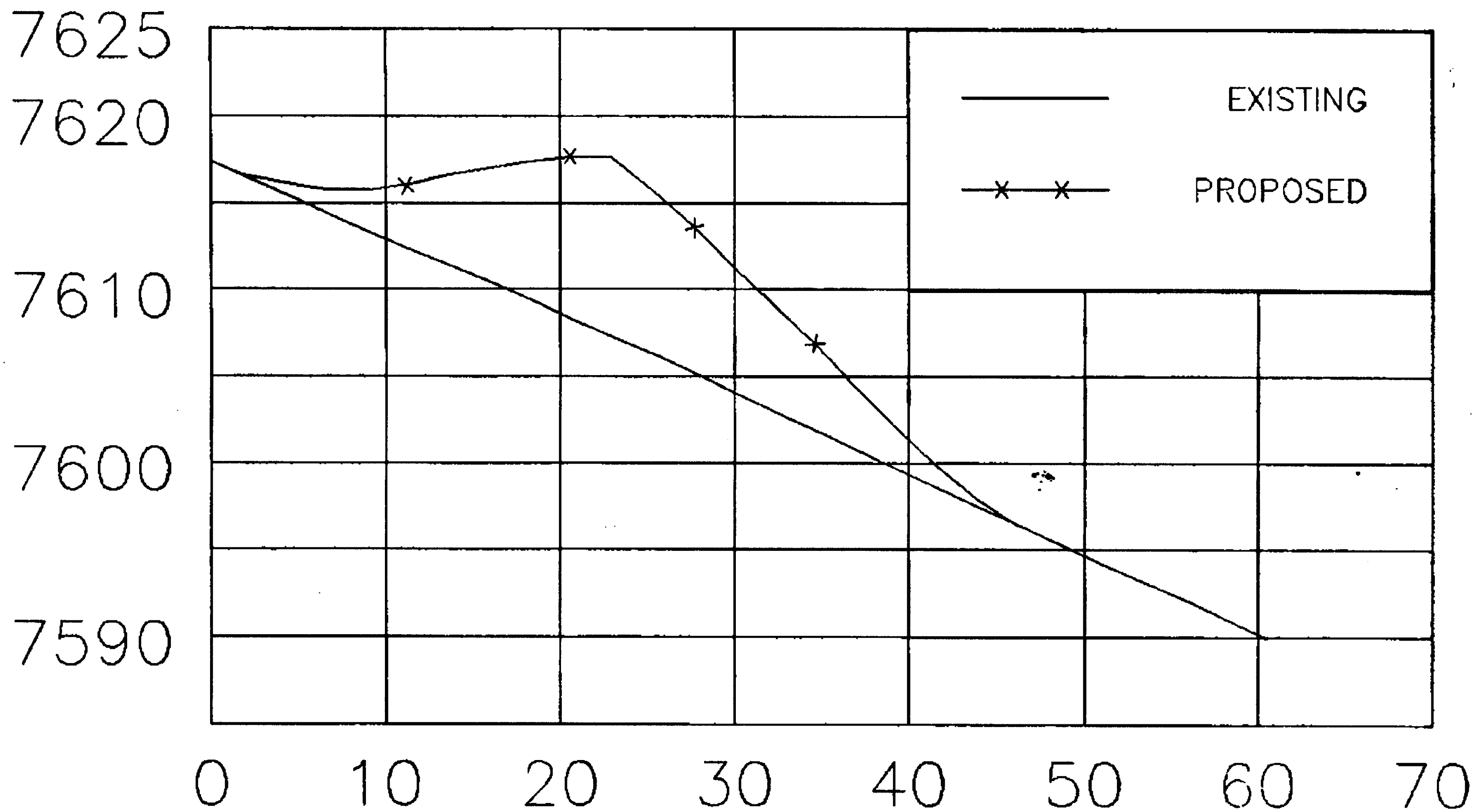

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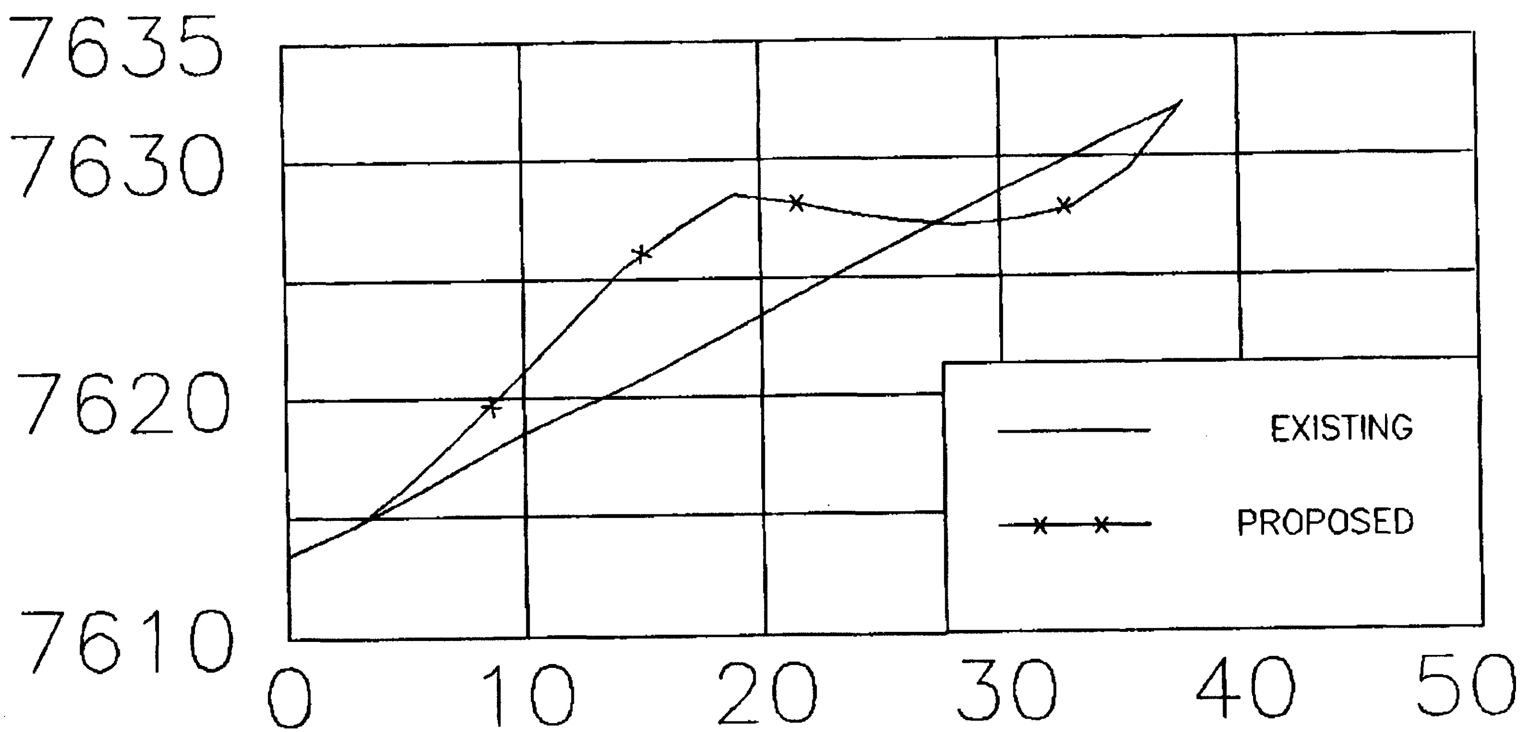

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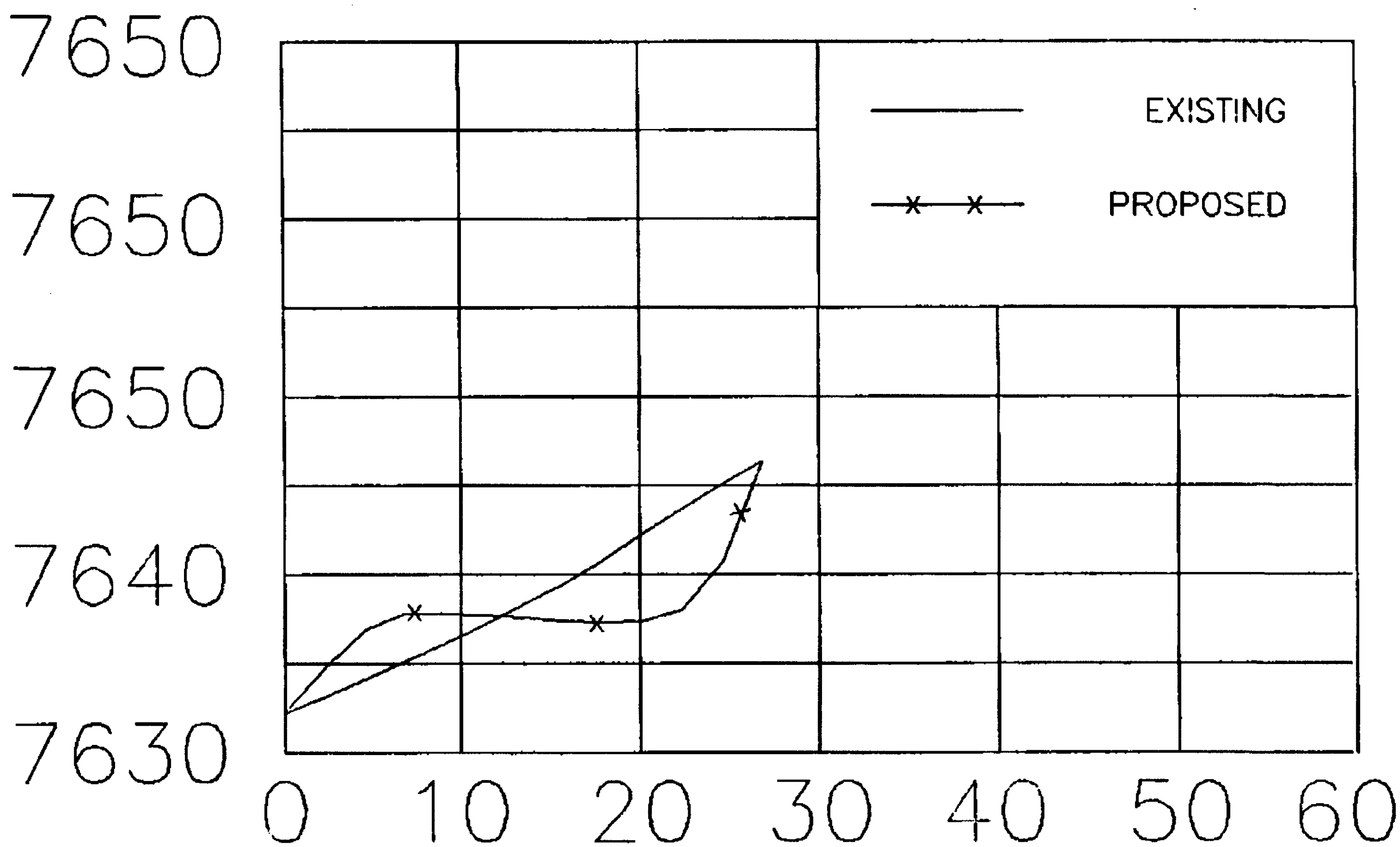

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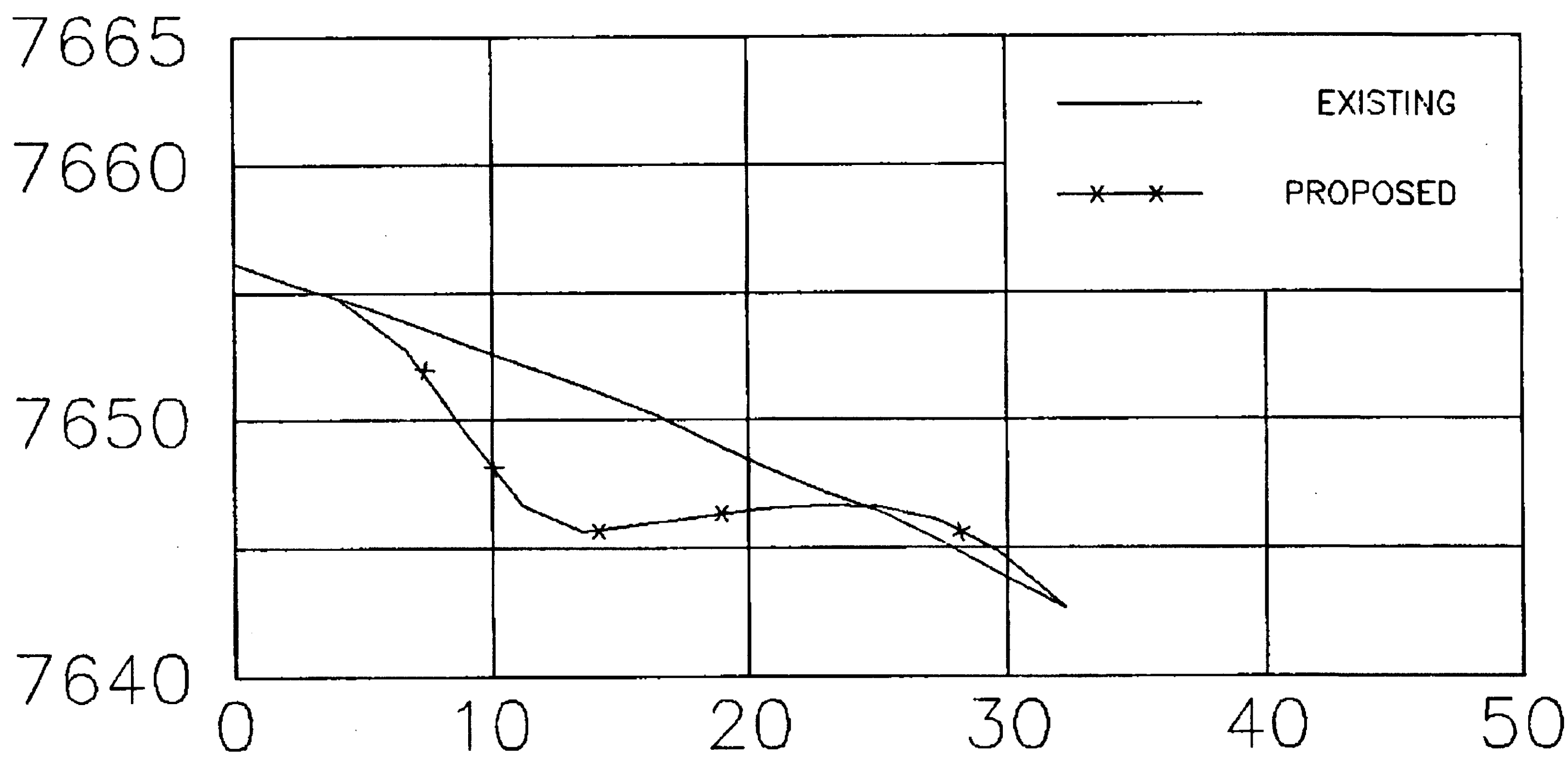

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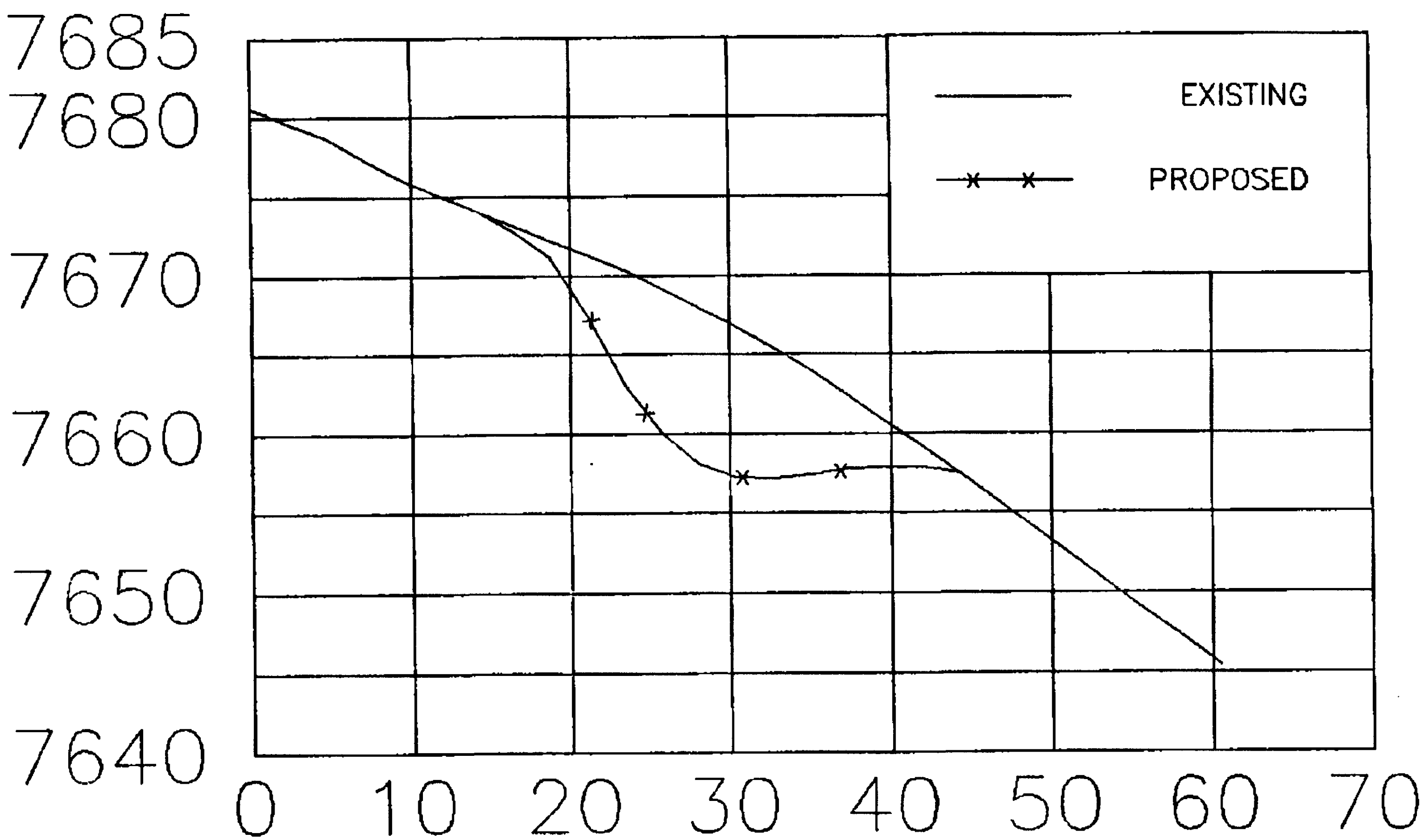

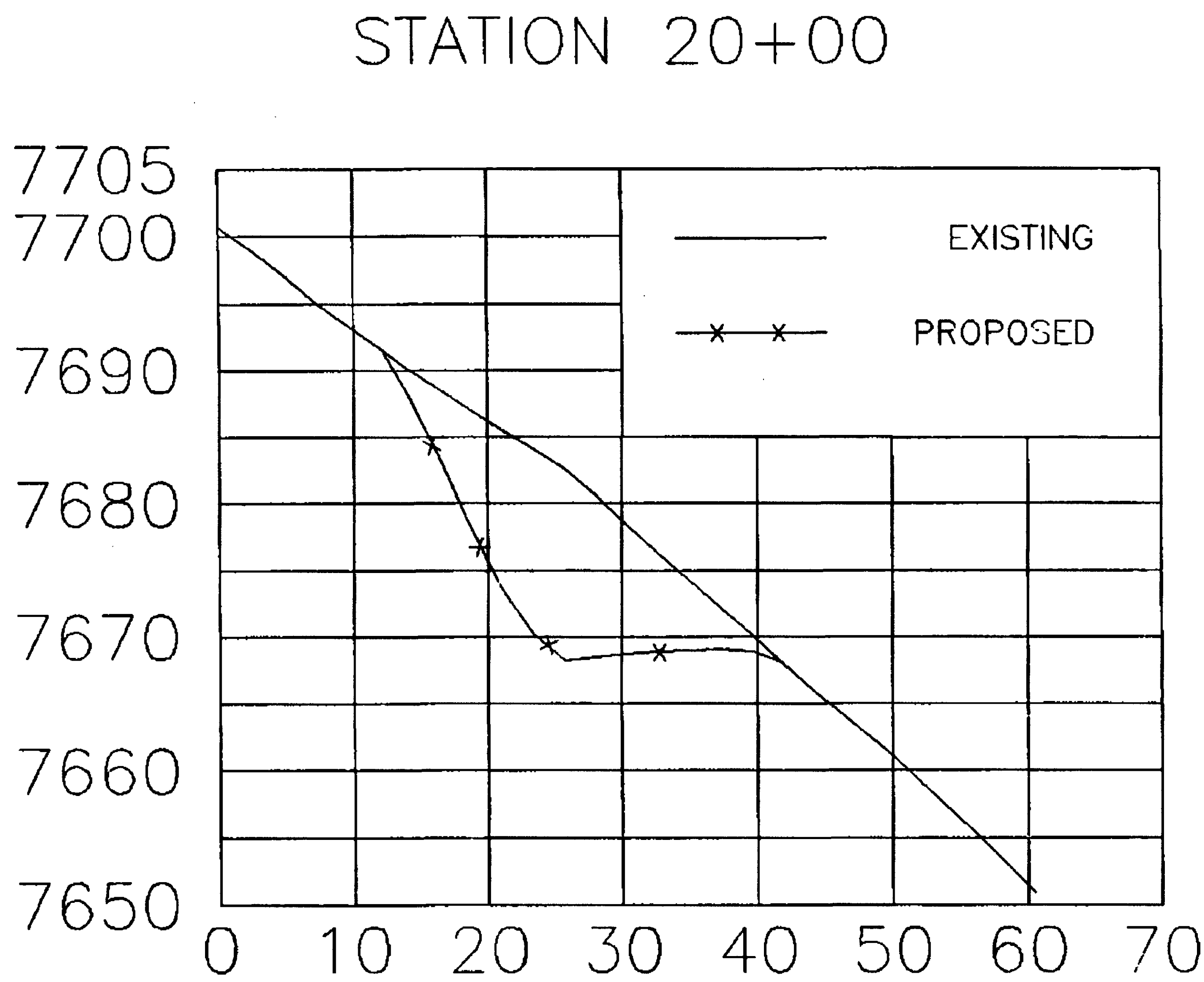
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PROPOSED
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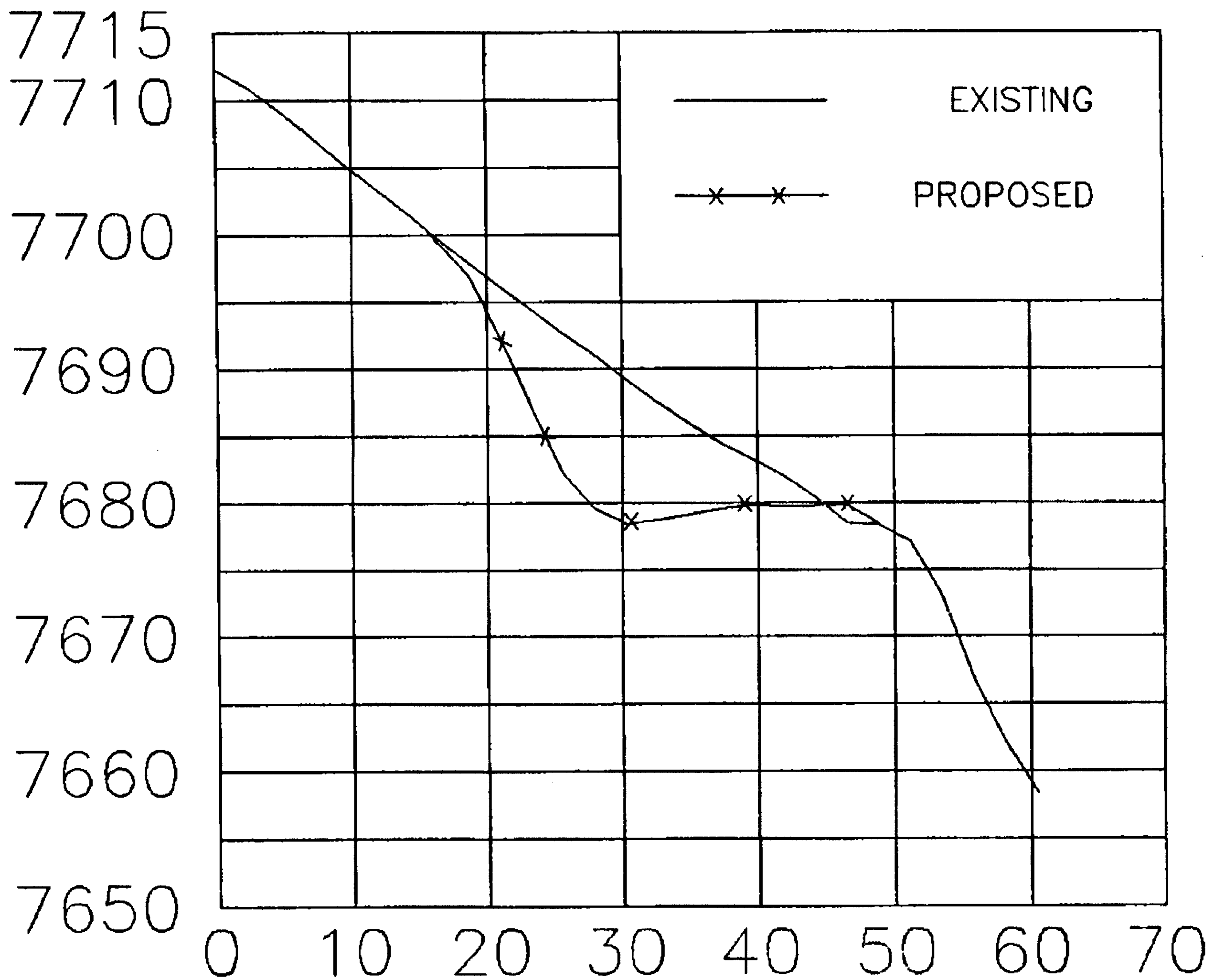

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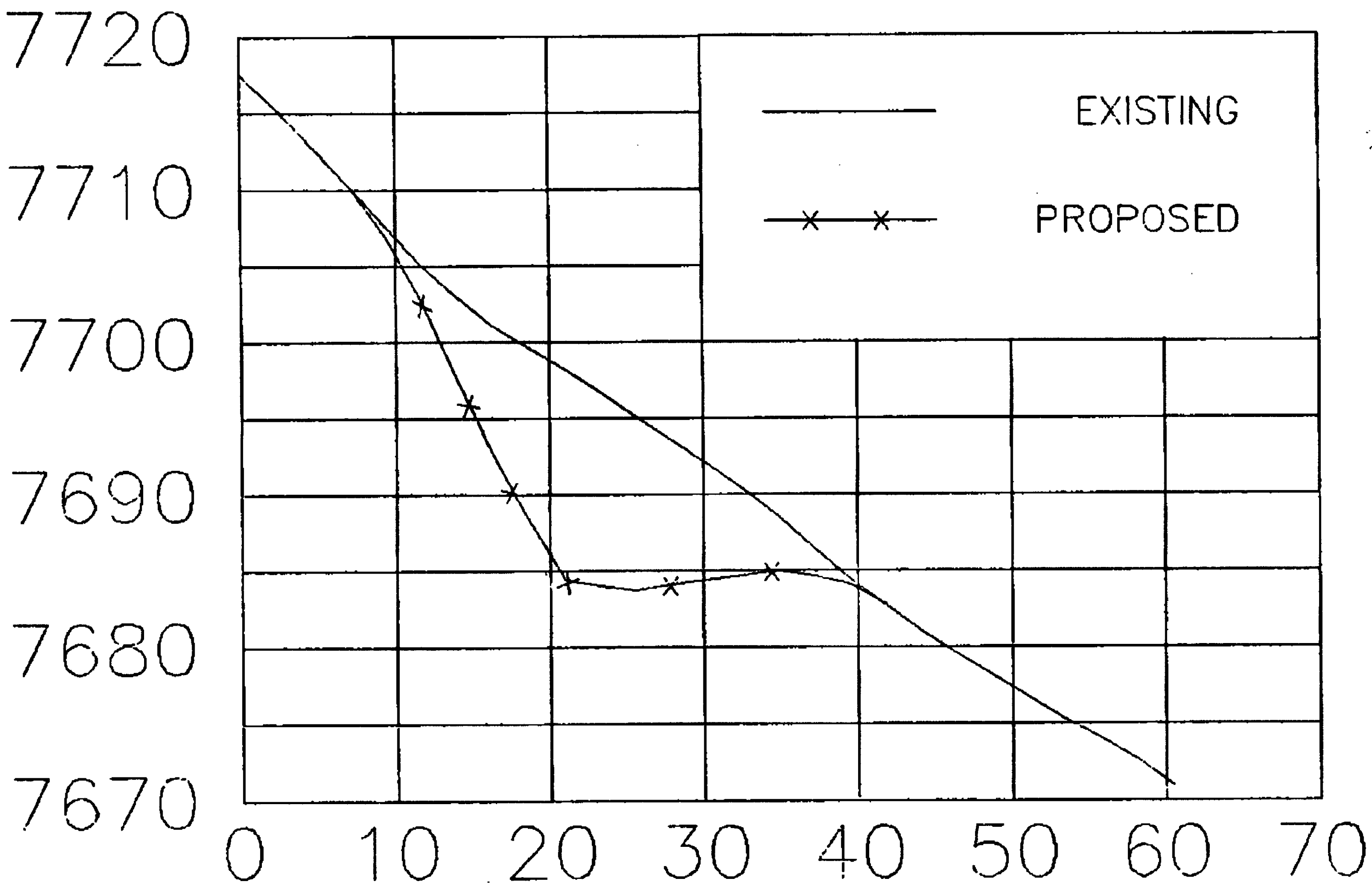

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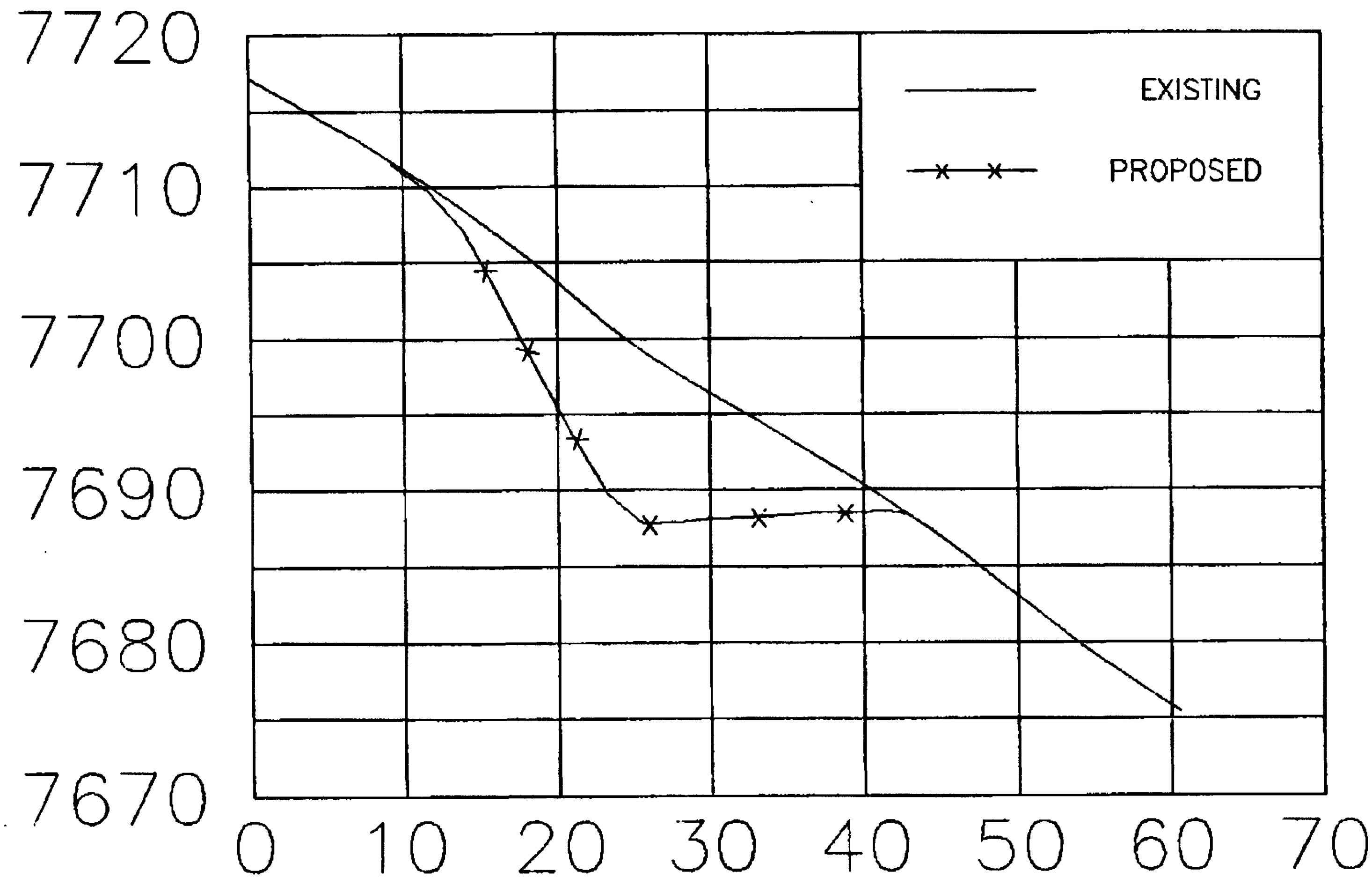

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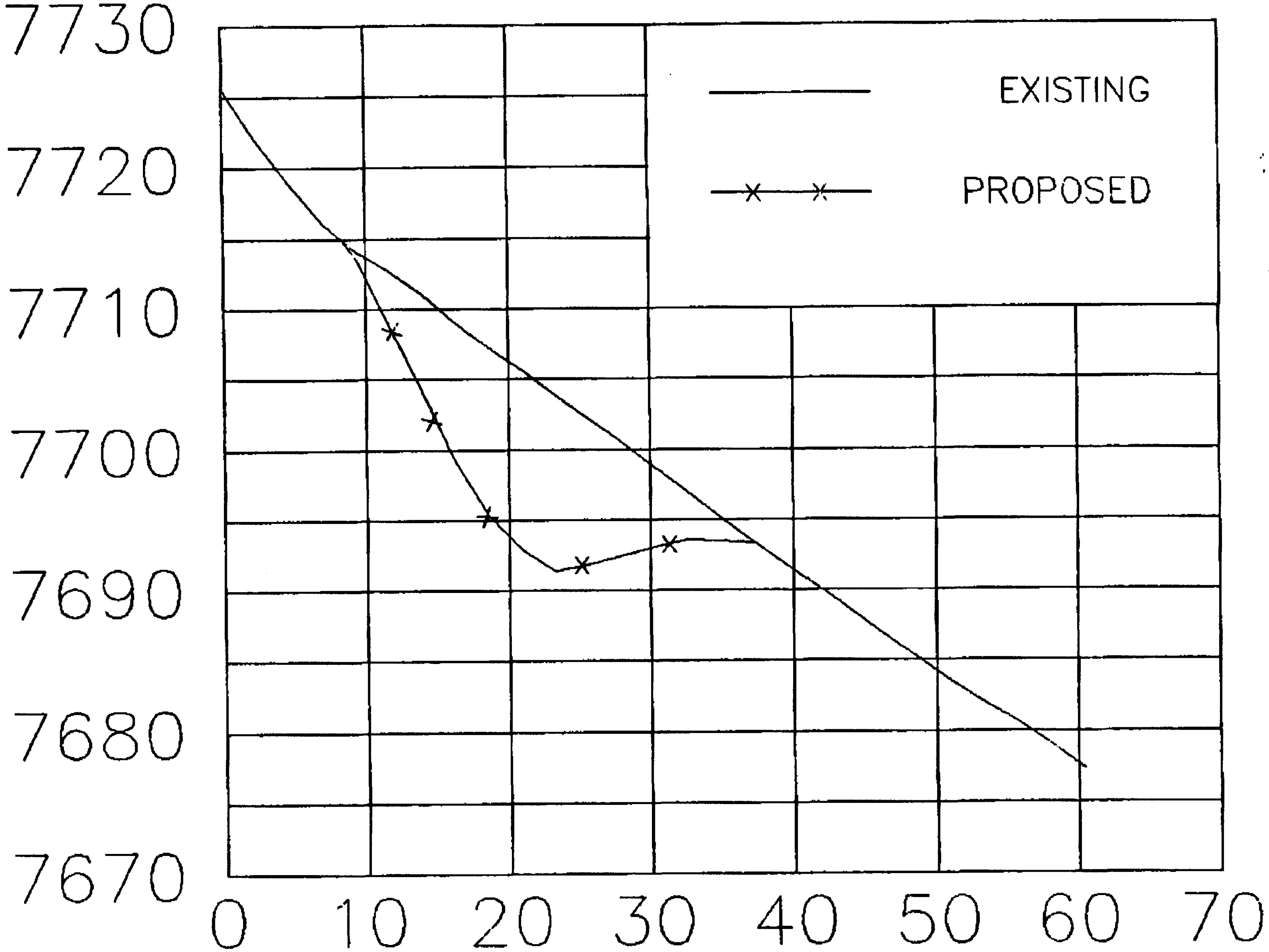

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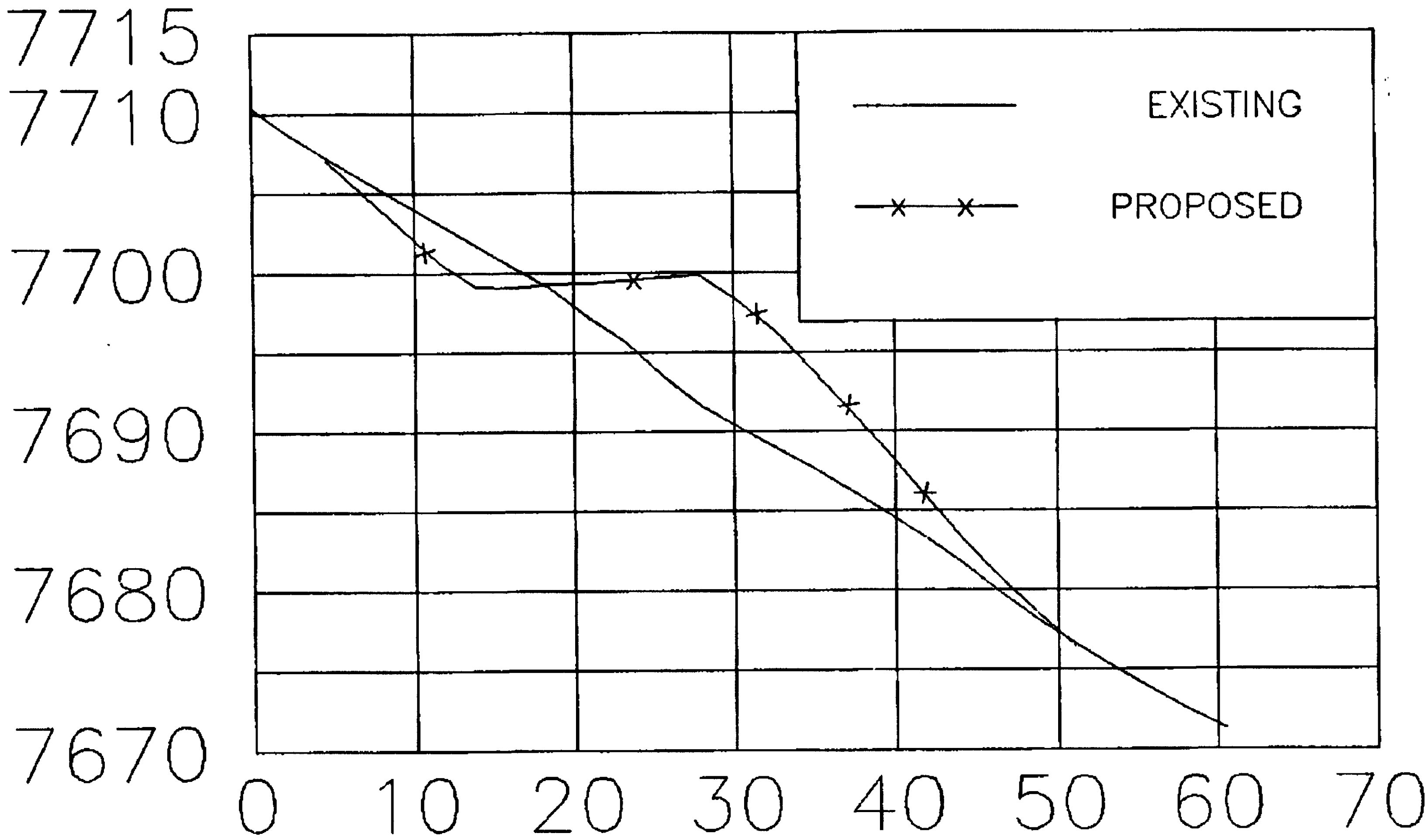

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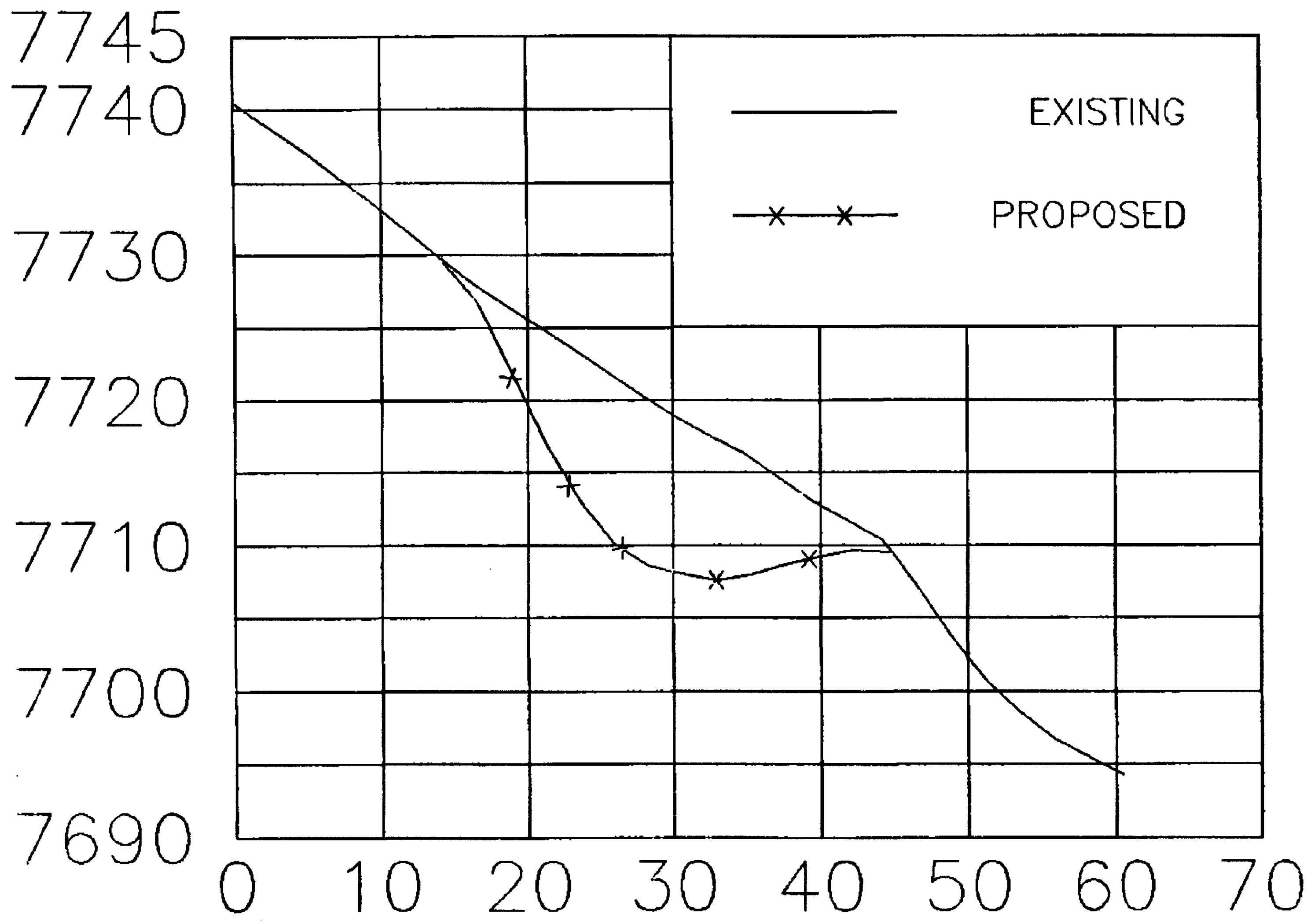

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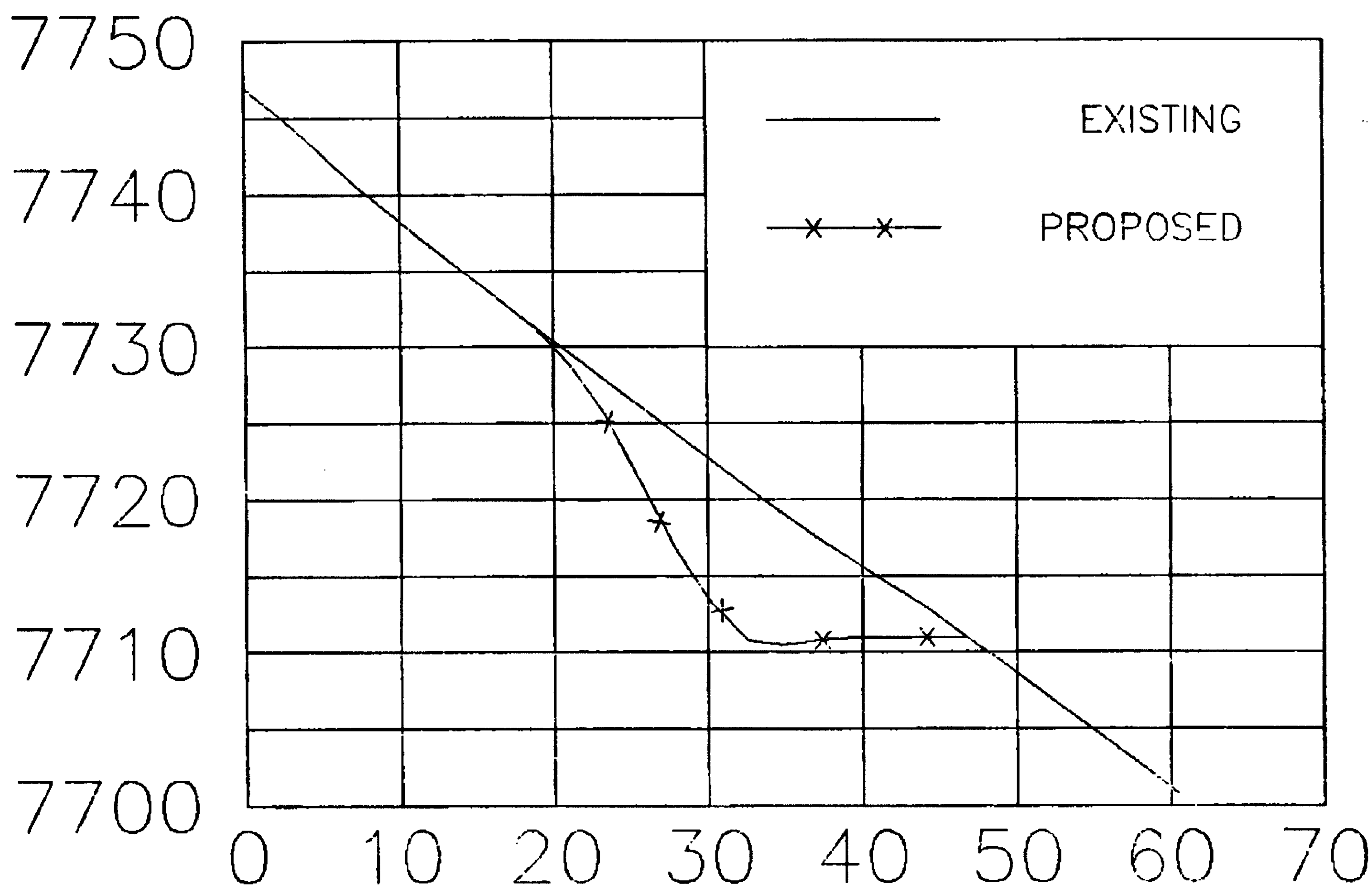

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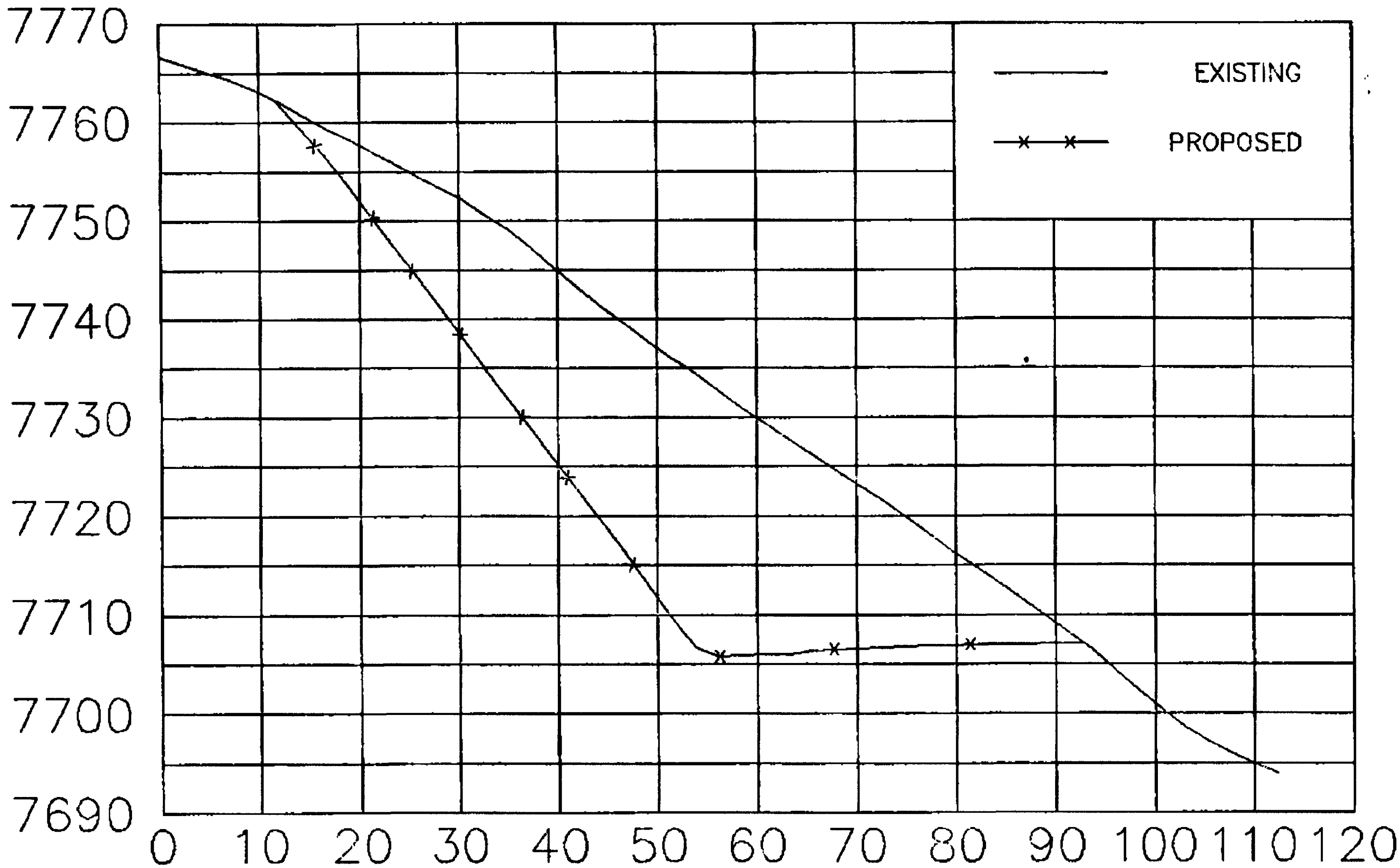

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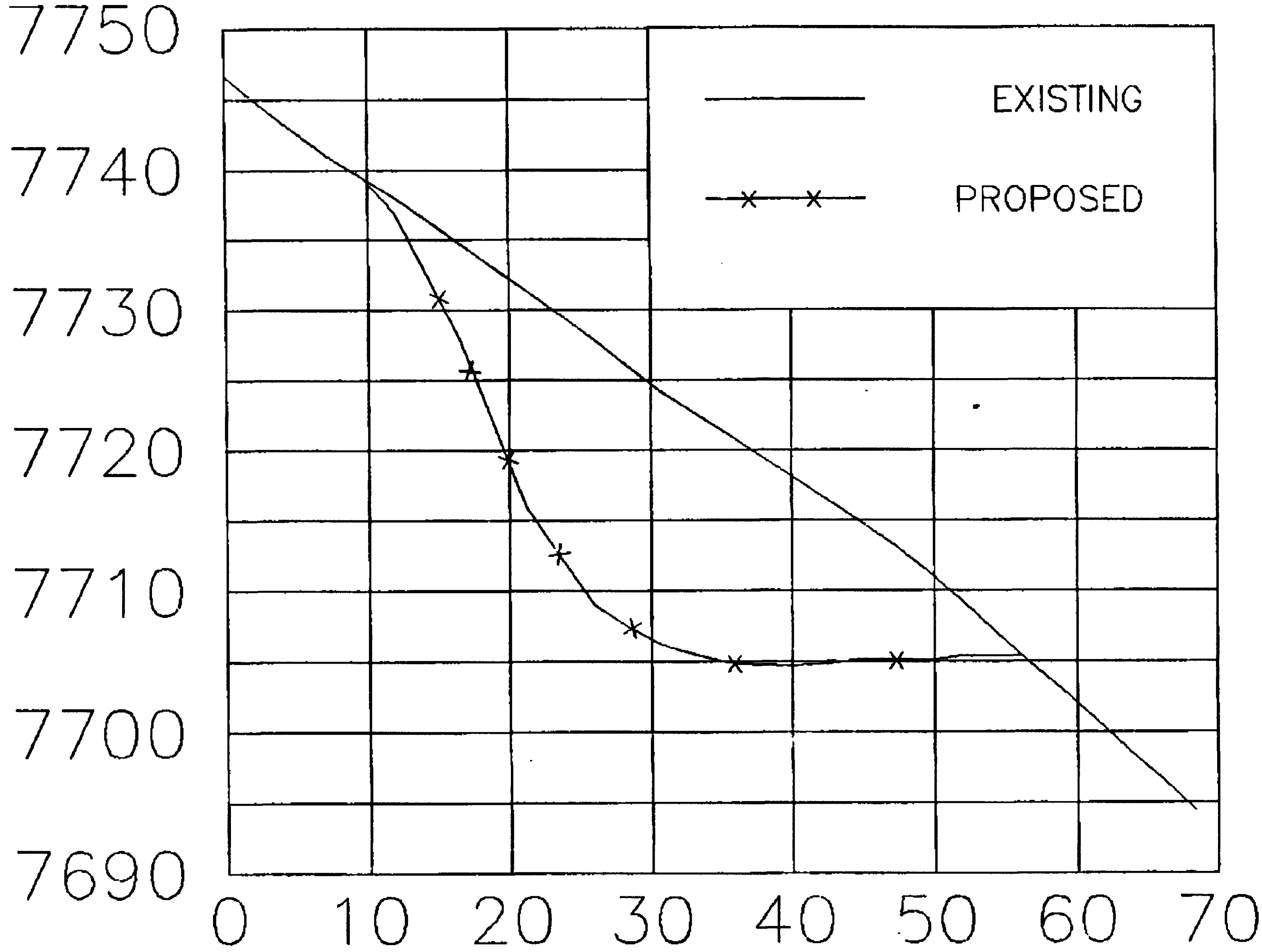

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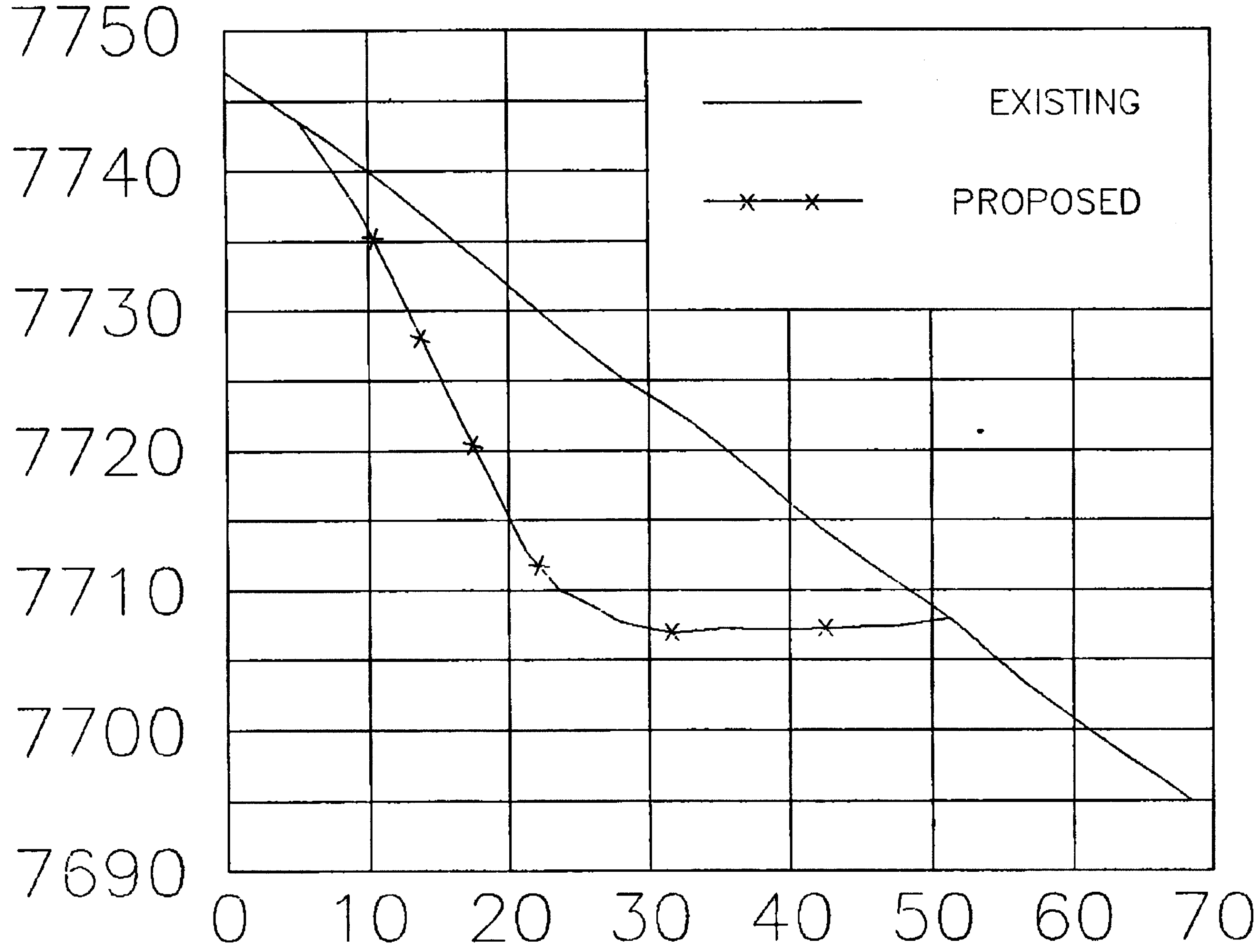


TANK SEAM ACCESS ROAD

SLOPE STABILITY ANALYSIS

DAMES & MOORE

127 SOUTH 500 EAST, SUITE 300, SALT LAKE CITY, UTAH 84102-1959
(801) 521-9255 FAX: (801) 521-0380

May 6, 1994

CO-OP Mining Company
P.O. Box 1245
Highway 31
Huntington Canyon
Huntington, Utah 84528

Attention: Mr. Charles Reynolds
Mining Engineer

Report
Geotechnical Consultation
Tank Seam Portal Access Road
Job No. 27437-001-162

INTRODUCTION

An initial report addressing the stability of the Tank Seam Portal Access Road, dated September 16, 1993 was previously sent to CO-OP Mining Company. Several initial concepts concerning road construction have been re-evaluated. This report updates the initial report and incorporates latest construction concepts.

CO-OP Mining Company proposes to construct an access road in Bear Creek Canyon to the Tank Coal Seam located at an elevation of approximately 7,720 feet. The general location of the proposed road relative to existing coal facilities is shown on Plate 1, Plot Plan.

On August 30, 1993, Wendell Owen and Charles Reynolds contacted Dames & Moore and requested that the design cut and fill slopes of the proposed extension of the existing access road at the Bear Creek Mine be evaluated. CO-OP Mining Company plans to commence underground mining of the Tank Seam which is above the Bear Creek Seam currently being mined. On September 2, 1993, an engineering geologist from the Salt Lake City office of Dames & Moore made a reconnaissance of the proposed access road alignment accompanied by Charles Reynolds, Project Mining Engineer for CO-OP Mining Company. Observations and recommendations resulting from that reconnaissance and subsequent analyses are presented in this report.

BACKGROUND

CO-OP Mining Company is in the process of expanding its underground coal mining operation and proposes to mine the Tank Coal Seam approximately 300 feet above the Bear Creek Seam currently being mined. To access the Tank Seam, CO-OP Mining Company proposes to extend the existing access road from the Bear Creek Portal, at elevation 7,440 feet, to the Tank Seam portal area, at an elevation of approximately 7,720 feet.

An engineering geologist and a geotechnical engineer from Dames & Moore visited the site in 1980 and performed a reconnaissance survey of the area prior to the construction of the Bear Creek Portal access road. That road was built with cut slopes in rocky slope wash (colluvium) using side-cast construction. The road has been serviceable with little or no maintènance problems since construction was completed. Two letter reports addressing the Bear Creek Portal Access Road cut and fill stability were provided to CO-OP Mining Company, dated December 29, 1980 and February 20, 1981.

The proposed access road to the Tank Seam Portal is expected to be 14 feet wide with a drainage ditch and safety berm. The total width may approach 20 feet. Planned construction is expected to be achieved through cut and fill methods.

SITE CONDITIONS

The general location of the Tank Seam Portal Access Road is shown on Plate 1, Plot Plan. The access road will be constructed by cut and fill methods. The average natural slope angle of the slopes across which the Tank Seam access road must traverse is approximately 35 degrees. The majority of material to be excavated is expected to consist of fine to coarse gravel, cobble, and boulder sized pieces of sandstone in a matrix of sand and clayey silt. This material generally has a slight to medium dry strength due in part to the presence of calcium carbonate cementation. Some natural slopes in this material are standing vertically up to a height of 15 feet.

Bedrock of moderately hard to hard, fine to medium grained, bedded sandstone is present as scattered outcrops throughout the slope. Most of the outcrops form prominent near horizontal ledges on the slopes. Vertical joints form boulder sized blocks in the bedrock which in turn produce angular boulder and cobble sized rock fragments that are scattered across the slope surface. Occasional juniper and pinon pine trees as well as sage brush and other large shrubs comprise the majority of vegetation on the slopes.

SOIL CONDITIONS

During the reconnaissance of the Tank Seam Portal Access Road, two soil samples were obtained to determine the engineering properties of the soil. The approximate locations where the soil samples were taken are shown on Plate 1, Plot Plan. The results of a partial grain size analysis are tabulated below:

SAMPLE NO.	SIEVE NO.	PERCENT FINER BY WEIGHT
1	# 4	91.6
	# 10	74.4
	# 40	62.7
	#200	27.1

DAMES & MOORE

SAMPLE NO.	SIEVE NO.	PERCENT FINER BY WEIGHT
2	# 4	74.0
	# 10	65.8
	# 40	55.3
	#200	37.0

Results of Atterberg limits tests performed on material finer than the number 200 sieve for both samples are tabulated below:

SAMPLE NO.	LIQUID LIMIT %	PLASTIC LIMIT %	PLASTICITY INDEX	UNIFIED SOIL CLASSIFICATION
1	24	19	5	CL-ML
2	22	16	6	CL-ML

These tests indicate that the finer fraction of the natural soil in the area is more silty than the soil tested previously during construction of the Bear Creek Portal Access Road.

Consolidated drained direct shear tests were performed on remolded portions of these two samples. The average strength values obtained from the direct shear tests indicate an effective stress friction angle of 32 degrees and a cohesion value of 180 psf. These soil properties were utilized in the subsequent slope stability analyses for both the cut slopes and the major fill slopes.

SLOPE STABILITY ANALYSIS

An analysis of the stability of proposed cut slopes and fill sections was performed utilizing a two dimensional, limit equilibrium stability program called PCSTABL5. For the cut slope evaluation, an automatic search routine was employed to determine the failure surfaces with the lowest factors of safety. Both random and circular failure surfaces were evaluated. For the fill section, a specified failure surface was input to represent the critical failure surface. For both analyses, a natural slope of 35 degrees was modeled considering dry conditions.

The analyses of the cut slope section considered vertical cuts, one horizontal to four vertical (1H:4V) cuts, and one horizontal to two vertical (1H:2V) cuts. Bedrock was conservatively modeled to be present 6 feet vertically below the ground surface and to trend parallel to the natural cut. The depth to bedrock along the Tank Seam Access Road is realistically expected to vary from exposures at the ground surface to depths of 2 to 3 feet. From a constructability standpoint, a 1H:2V cut configuration was determined to be the most stable. Plate 2 presents the configuration modeled, the input material properties, and the ten failure surfaces with the minimum safety factors. A minimum factor of safety of 1.4 was determined for the input configuration. The "arrows" on Plate 2 indicate the failure surface with the lowest safety factor.

The analyses of the fill section modeled an access road entirely constructed on fill. The fill slope was input with a 1H:1V slope and a specified failure surface was evaluated. Plate 3 presents the slope configuration, input properties, and the minimum safety factor. The natural slope was modeled as a plane surface, however, in reality this surface would be stepped due to occasional bedrock outcrops, and the factor of safety may actually be somewhat higher than portrayed.

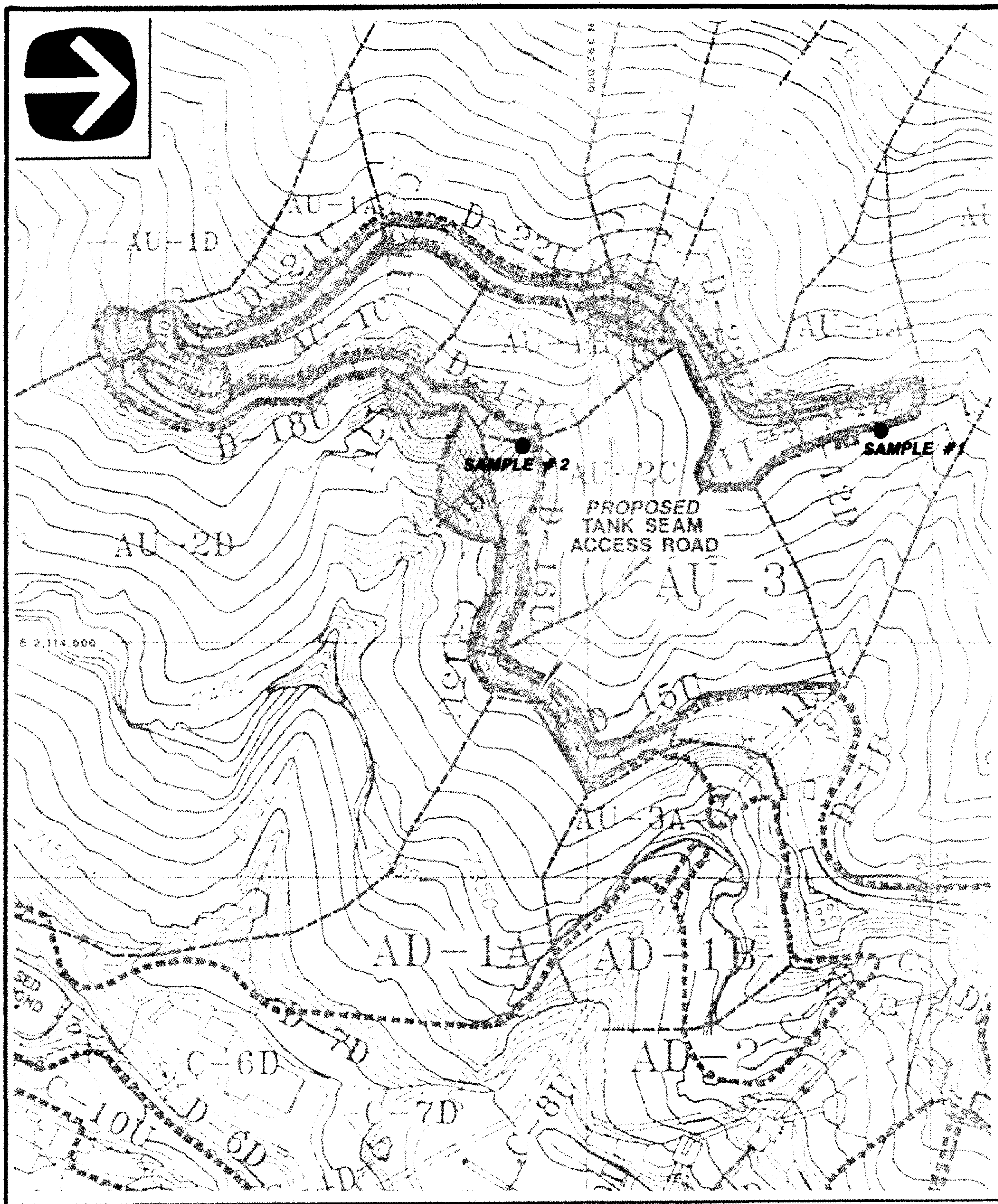
CONCLUSIONS AND RECOMMENDATIONS

It is our understanding, from a telephone conversation with Charles Reynolds, that a minimum factor of safety required for cut and fill slopes along the access road is 1.3. The computer analysis for cut slope stability indicates a minimum static factor of safety of 1.4 for a slope of 1H:2V, or a slope angle of approximately 63 degrees. In the stability analysis we conservatively assumed a depth to bedrock of 6 feet. The depth to bedrock along the Tank Seam Access Road alignment will probably vary from exposures at the ground surface to depths of 2 to 3 feet. Only in the larger gullies, where the road will be mostly or entirely on fill, will the depth to bedrock exceed the assumed average of 6 feet. The effect of the calcium carbonate cementation in the soil was not incorporated into the analysis. If it had been, the factor of safety would be significantly higher.

The large fill across the gully where soil sample No. 2 was taken is expected to have a factor of safety of approximately 1.4 if the fill slope is constructed at a slope of 1H:1V. The fill material should be compacted in lifts not exceeding 18 inches in depth and compacted by heavy machinery during placement. All rock fragments in excess of 18 inches should be removed from the fill. Rock fragments incorporated in the fill should be placed in a manner to minimize void space. It is recommended that the natural slope surface be prepared for the fill by removing vegetation and loose cobble and boulder sized rock fragments. Cobble and boulder sized rock fragments securely embedded into existing slopes may be left in-place provided adequate compaction is achieved adjacent to these fragments. We recommend that a series of narrow terraces, approximately 10 feet wide, be constructed to key the fill material into the natural slopes.

Stability of the slopes will be influenced by the degree of saturation of existing soils. Therefore, surface drainage must be channeled to prevent or at least minimize runoff over the slopes. Also, snow removed from the access road should be placed at the south end of the switchback to minimize the amount of moisture percolating into the fill slopes or road surface due to melting snow. We understand that erosion control mats will be placed on constructed slopes to minimize minor slides and sluffs.

We understand that boulders obtained from cut areas during construction, that are not incorporated in fill areas, will be placed on terraced portions of fill slopes. Boulders will be placed in a single lift on horizontal terraces to simulate natural conditions.



100 0 100 200
SCALE IN FEET

AD-1 INDICATES DRAINAGE AREAS THAT ARE DIVERTED THROUGH SEDIMENT PONDS OR ALTERNATE SEDIMENT CONTROL STRUCTURES

AU-1 INDICATES DRAINAGE AREAS THAT ARE DIVERTED BY DITCH AND/OR CULVERT TO OFF-SITE WITH LIMITED OR NO SEDIMENT CONTROL

REFERENCE-
ADAPTED FROM DRAWING ENTITLED, "WATERSHED MAP; BEAR CANYON," BY CO-OP MINING CO., HUNTINGTON, UTAH, PLATE 7-5, DATED 5-6-91.

CO-OP MINING CO.

PLOT PLAN

BEAR CANYON MINE

Dames & Moore

PLATE 1

FILE 27437-002-162 BY ____ DATE ____ CHECKED BY ____ DATE ____

DAMES & MOORE

Based on the slope stability analysis and observations made during the reconnaissance visit, it is our opinion that the cut and fill slopes will perform satisfactory and similar to those located along the Bear Creek Portal Access Road.

We appreciate the opportunity of visiting the site and assisting you with the expansion of the Bear Creek Mine. If you have any questions concerning this report or if we can assist you in any other way please call at your earliest convenience.

Sincerely,

DAMES & MOORE, INC.

C. Charles Payton

C. Charles Payton, C.E.G.
Associate

Russell L. Owens

Russell L. Owens, P.E.
Professional Engineer
State of Utah

RUSSELL L. OWENS

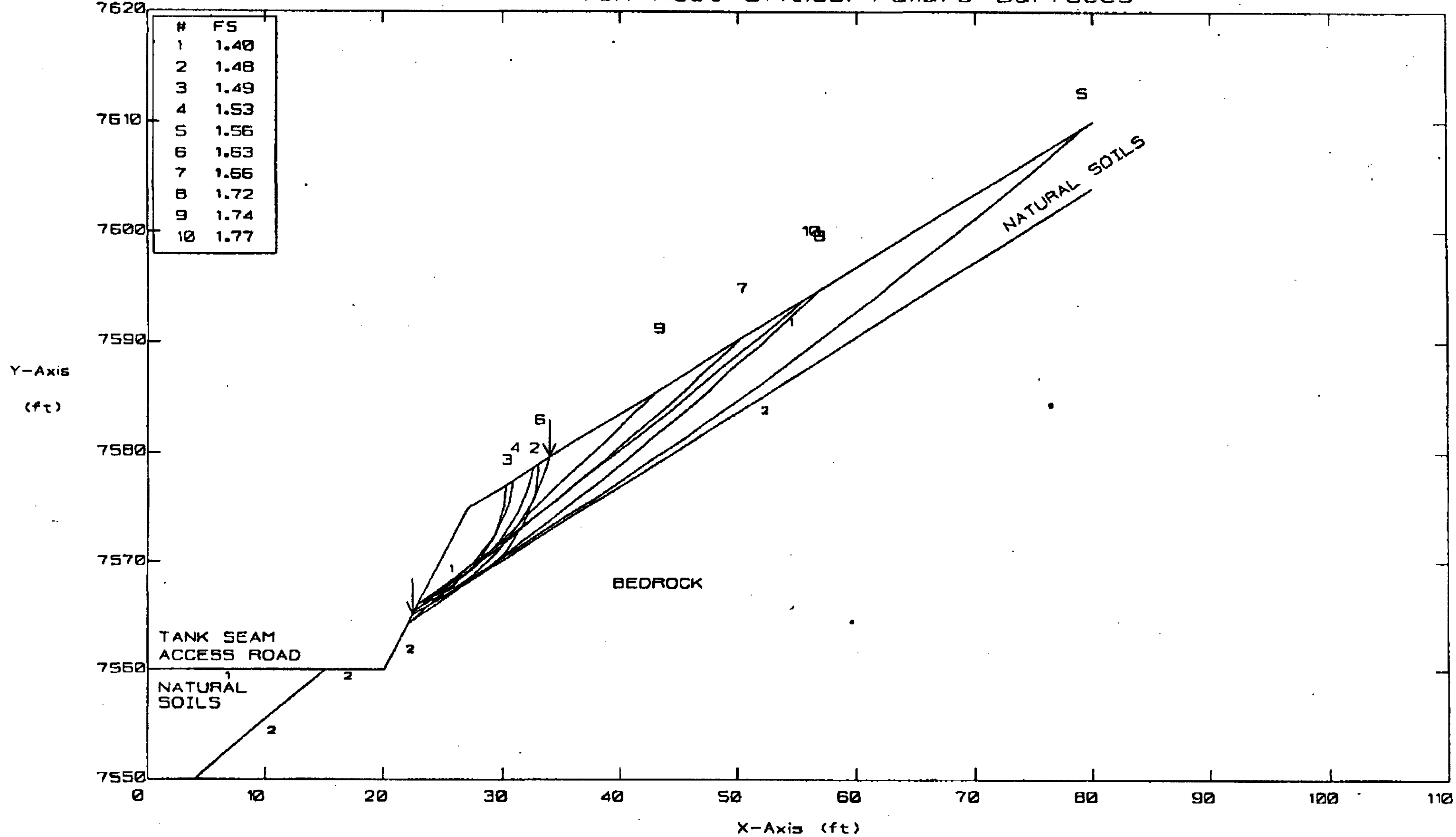
Attachments:

Plate 1 - Plot Plan
Plate 2 - Cut Slope Cross-Section
Plate 3 - Fill Slope Cross-Section

FILE ______ BY ______ DATE ______ CHECKED BY ______ DATE ______

CO-OP Mine 1H:2V Cut slope

Ten Most Critical Failure Surfaces



Soil Type No.	Total Unit Wt. (pcf)	Saturated Unit Wt. (pcf)	Cohesion Intercept (psf)	Friction Angle (deg)
1	120	125	180	32
2	130	140	500	38

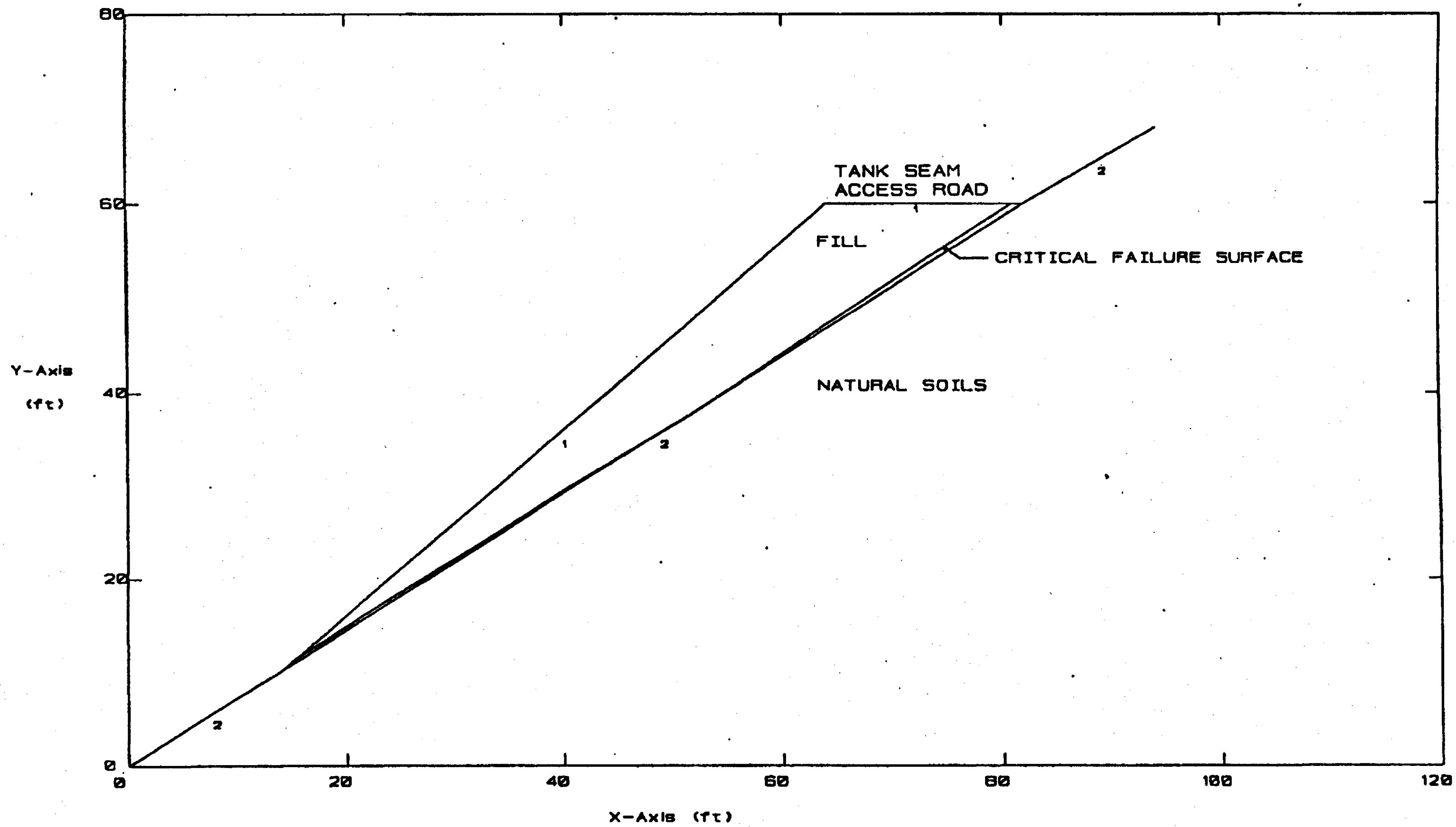
Dames & Moore

PLATE 2

FILE ______ BY ______ DATE ______ CHECKED BY ______ DATE ______

CO-OP FIII 1:1 FILL SLOPE

Specified Critical Failure Surface, Minimum FS=1.44



Soil Type No.	Total Unit Wt. (pcf)	Saturated Unit Wt. (pcf)	Cohesion Intercept (psf)	Friction Angle (deg)
1	125	130	100	36
2	120	125	100	32

Dames & Moore

PLATE 3

DAMES & MOORE

127 SOUTH 500 EAST, SUITE 300, SALT LAKE CITY, UTAH 84102-1959
(801) 521-9255 FAX (801) 521-0380

July 12, 1994

CO-OP Mining Company
P.O. Box 1245
Highway 31
Huntington Canyon
Huntington, Utah 84528

Attention: Mr. Charles Reynolds

RE: Clarifications to Tank Seam Portal Access Road Report

Dear Charles,

The Division of Oil, Gas, and Mining has asked for two clarifications concerning the Tank Seam Portal Access Road report, dated May 6, 1994. These clarifications include the potential for failure surfaces within the natural soils underlying fill areas of the Tank Seam access road and the use of 18 inch lifts for road construction.

The depth to bedrock along the Tank Seam Access Road is expected to vary from exposures at the ground surface to depths of 2 to 3 feet. The bedrock outcrops and soil deposits along the access road form a "stepped" slope. Plate 3 of the report presents a simplified natural slope in order to determine the factor of safety for failure surfaces within the fill. The natural slope was portrayed as a relatively smooth soil slope for ease of modelling since the potential of a failure surface extending through the natural slope, as opposed to failure within the fill, was considered remote. A model portraying a "stepped" slope with appropriate bedrock shear strength properties would confine critical failure surfaces within the road fill.

Dames & Moore's September 16, 1993 report recommended that the fill material utilized for the Tank Seam Access Road construction "be compacted in lifts not exceeding 8-inches in depth and all rock fragments cobble sized and larger should be removed from the fill". This recommendation was changed in our May 10, 1994 such that 18 inch lifts could be utilized and that rock fragments in excess of 18 inches could be incorporated into upper lifts provided the material adjacent to the rock fragments was properly compacted. The original recommendation of 8-inch lifts and removal of cobble size fragments is standard for structural fill underlying foundations to assure adequate bearing capacity and to minimize settlement. Since bearing capacity and minor settlement are not issues with the access road, we felt that 18 inch lifts would not reduce safety factors for the fill and would expedite construction provided the lifts were properly compacted to minimize void spaces.

We hope these clarifications are beneficial. If additional information is required, please do not hesitate to call.

Respectfully,

DAMES & MOORE, INC.

Russell L. Owens

Russell L. Owens, P.E.
Senior Geotechnical Engineer

Tank Seam Fill

As Constructed

Slope Stability Analysis

Introduction

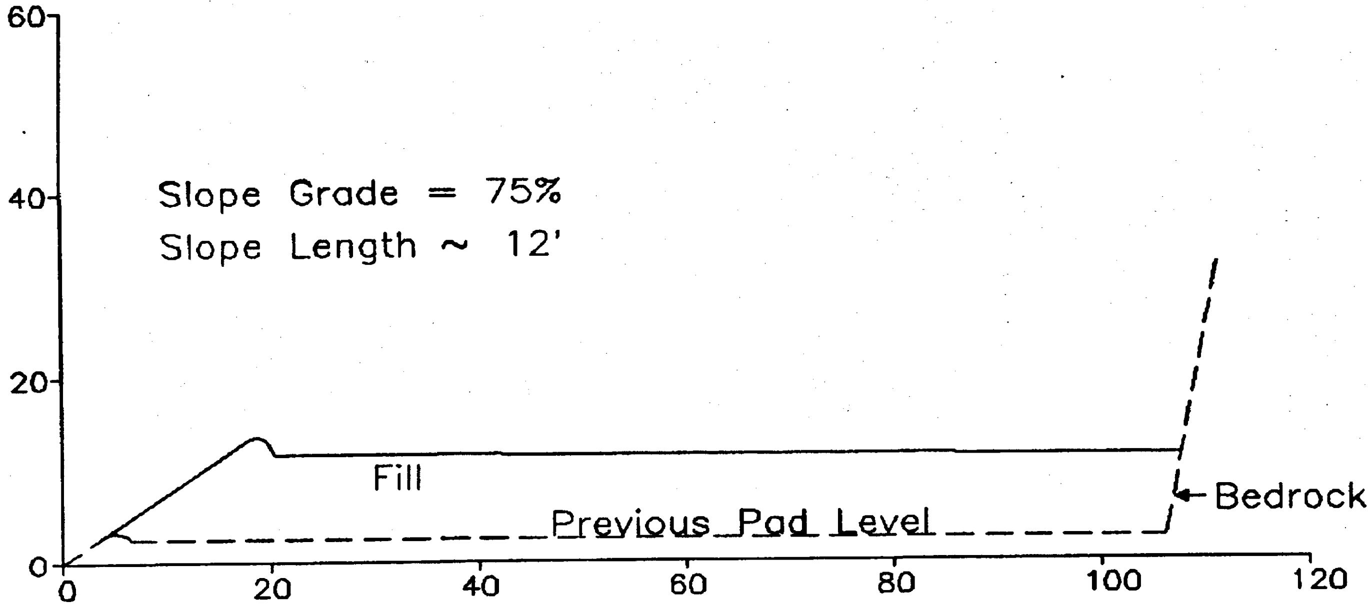
The following section contains slope profiles and a stability analysis for each fill slope along the Tank Seam Access Road. Final backfilling and grading along the access road was completed in November 1994.

Soil samples for the stability analysis were collected from the fill material prior to placement of the fill. Analysis of the soil samples were performed by Dames & Moore. The slope stability analysis and safety factors were generated by Dames and Moore.

The location of the five fill areas are as follows (Stations are shown in Figure 5G-4). TSF-1 fill is located on the upper storage pad in the area of station 1+00. TSF-2 fill is located in the area of station 8+00 and 9+00 (large fill areas). TSF-3 fill is located between stations 10+00 and 11+00. TSF-3 fill is located at the switchback, in the area of Station 15+00. TSF-5 fill is located in the area of Station 25+00.

The following figures show detailed as constructed slope profiles of each fill area. Following the profiles is a slope stability analysis, which analyzes each fill area.

TSF-1

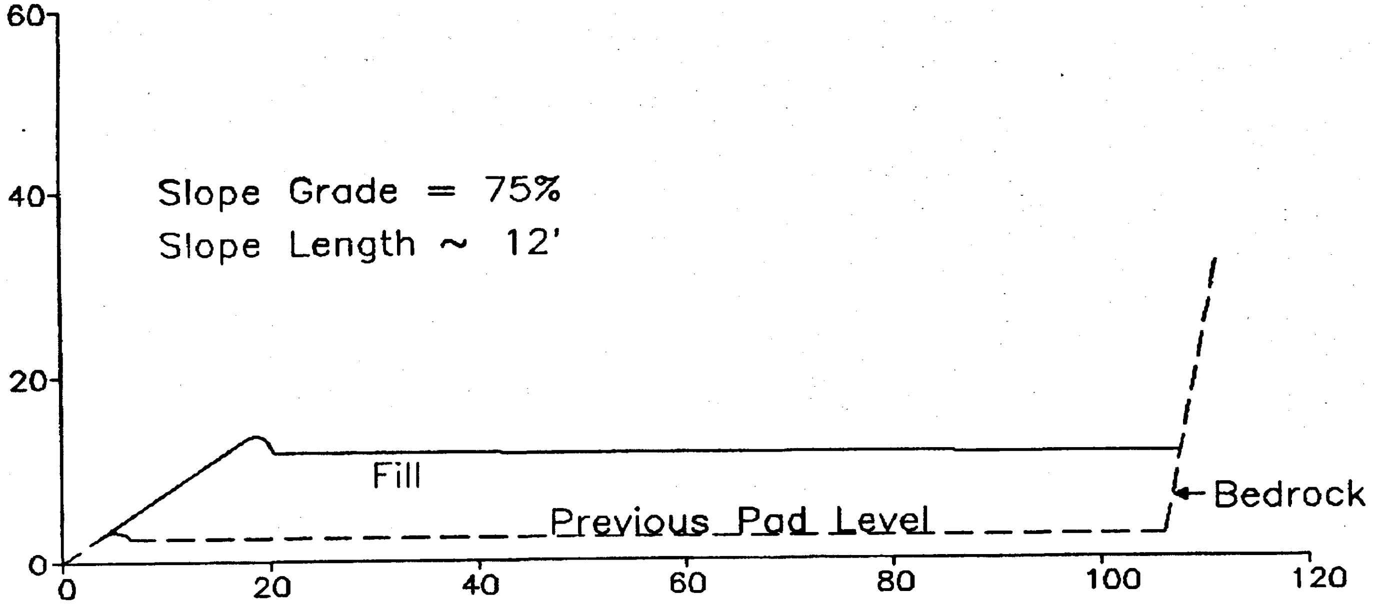


Constructed Slope ————————

Fill Base - - - - - - - - - - - -

Scale: 1" = 20'

TSF-1

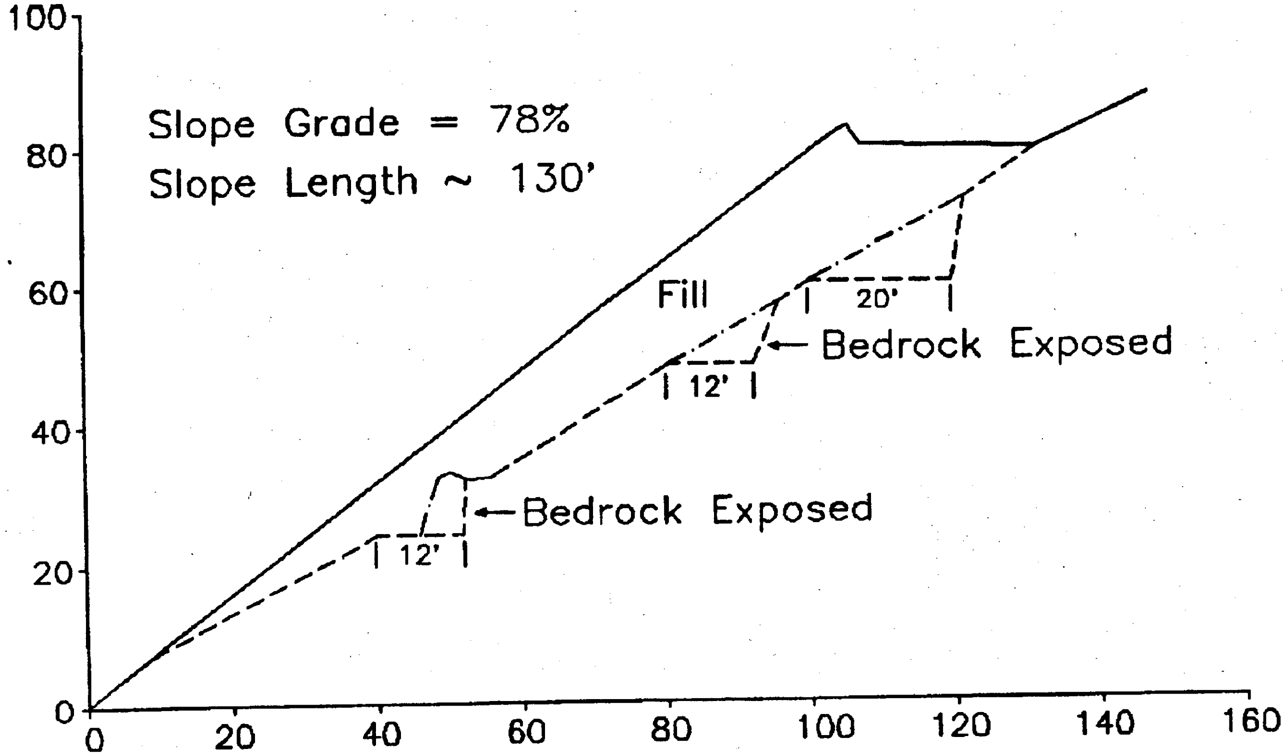


Constructed Slope ————————

Fill Base – – – – – – – – – – – –

Scale: 1" = 20'

TSF-2



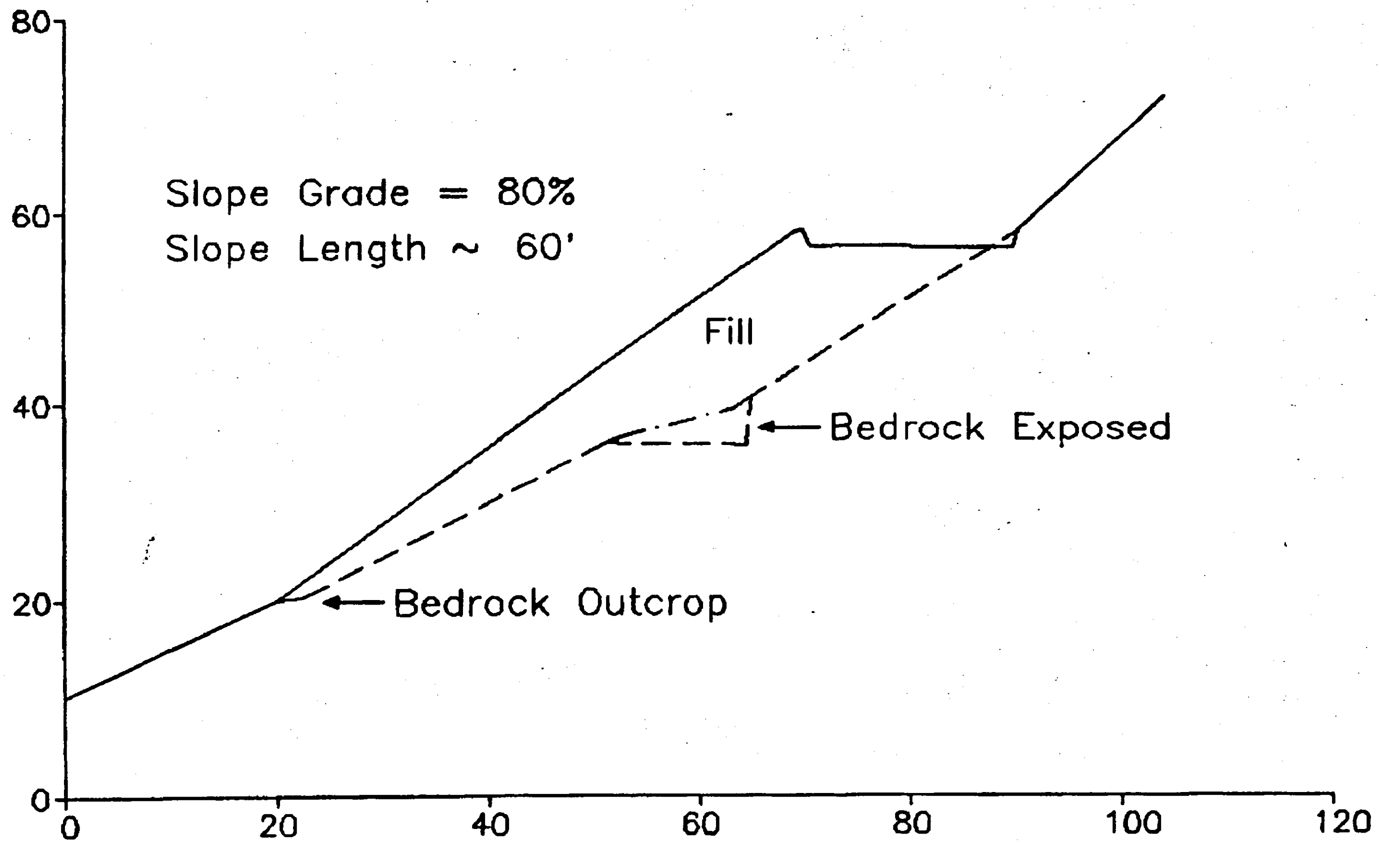
Constructed Slope

Base of Fill

Natural Slope
(where different from fill base)

Scale: 1" = 30'

TSF-3



Constructed Slope ————————

Fill Base — — — — — — — —

Natural Slope – · – · – · – · – · –
(where different from fill base)

Scale: 1" = 20'

DAMES & MOORE

127 SOUTH 500 EAST, SUITE 300, SALT LAKE CITY, UTAH 84102-1959
(801) 521-9255 FAX: (801) 521-0380

December 5, 1994

CO-OP Mining Company
P.O. Box 1245
Highway 31
Huntington Canyon
Huntington, Utah 84528

Attention: Mr. Charles Reynolds
Mining Engineer

Report
Geotechnical Consultation
Tank Seam Portal Access Road
Job No. 27437-001-162

INTRODUCTION

This report presents the results of stability analyses performed by Dames & Moore for Tank Seam Portal Access Road fill sections for CO-OP Mining Company.

PURPOSE & SCOPE OF STUDY

The purpose and scope of this study was defined in our proposal dated October 20, 1994. In accomplishing the work the following services were performed.

1) Analyzing laboratory tests to determine appropriate estimates of soil strength parameters to be utilized in the subsequent analyses.

2) Modeling five separate fill slope sections. "As built drawings of the slopes were supplied by CO-OP mine.

3) Compiling data into this document that summarizes laboratory tests data and analyses results.

LABORATORY DATA

The laboratory data used in our engineering analyses were obtained from mechanical grain size analyses, compaction tests, and consolidated drained direct shear tests. Initial results for sample TSF-3 seemed low, consequently a second sample, designated TSF-6, was collected and tested. Results of the two tests were averaged and used for modeling section 3. The test results are summarized in the following paragraphs.

MECHANICAL ANALYSES

To aid in classifying the soils mechanical sieve analyses were performed on bulk samples collected from fill that was to be placed at each fill section. Results of the gradation analyses are presented in Table 1.

Table 1
Mechanical Analyses Results

Sample Number	USCS Classification	Percent Gravel	Percent Sand	Percent Fines
TSF-1	SC-SM	23.9	43.9	32.2
TSF-2	SC-SM	25.4	35.0	39.6
TSF-3	SC-SM	25.2	27.8	47.0
TSF-4	CL-ML	17.4	23.2	59.4
TSF-5	CL-ML	17.6	19.2	63.2
TSF-6	SC-SM	26.7	30.1	43.2

COMPACTION TESTS

Compaction tests were performed on bulk samples collected from fill material that was to be placed at each fill section. Results of the gradation analyses are presented in Table 2.

Table 2
Compaction Test Results

Sample Number	USCS Classification	Optimum Moisture Content %	Maximum Dry Density PCF
TSF-1	SC-SM	10.3	124.4
TSF-2	SC-SM	7.9	131.7
TSF-3	SC-SM	6.3	139.7
TSF-4	CL-ML	9.2	128.9
TSF-5	CL-ML	9.6	132.4
TSF-6	SC-SM	7.8	136.6

DIRECT SHEAR TESTS

Direct Shear tests (consolidated drained) were performed on remolded bulk samples which were collected from fill that was to be placed at each of the fill sections. Samples were compacted to 95% of the maximum dry density as determined in the compaction tests. In accordance with ASTM 3080 the plus 3/8 material is removed prior to remolding. The plus 3/8 inch fraction of the samples ranged from 12.6 to 20.5 percent. Subjective interpretation was necessary for some tests due to scatter in the data points. The results are summarized in Table 3.

Table 3
Direct Shear Tests Results

Sample Number	USCS Classification	Friction Angle	Cohesion psf
TSF-1	SC-SM	38.7	168
TSF-2	SC-SM	28.0	290
TSF-3	SC-SM	36.6	0
TSF-4	CL-ML	40.6	15
TSF-5	CL-ML	28.2	510
TSF-6	SC-SM	36.5	162

SLOPE STABILITY ANALYSES

Analyses of the stability of the proposed fill sections were performed using a two-dimensional, limit equilibrium stability program called PCSTABL6. An automatic search routine was employed in each case to determine the failure surfaces with the lowest factors of safety. In all cases the fill was modeled assuming unsaturated conditions.

The geometry of each section was modeled based on the "as built" drawing provided by CO-OP. In each case 95% of the maximum dry density as determined by the compaction tests was input for the soil unit weight. Friction angles and cohesion were obtained from the direct shear test results. Bedrock strength properties were input to reflect significantly higher strength values than the fill material in order to determine the factors of safety of the fills.

STABILITY ANALYSES RESULTS

The results of our analyses show factors of safety in acceptable ranges. For sections 1 and 5, the minimum calculated safety factors were above 2.0. For section 2 the minimum calculated safety factor was 1.33. For section 3 the minimum calculated safety factor was 1.36. For section 4 the minimum calculated safety factor was 1.50.

It is our opinion that the fill slopes should generally perform satisfactorily. However, due to limited sampling and testing and potential variabilities in fill placement, we recommend periodic inspection of the fill slopes for any signs of distress, particularly at sections 2 and 3, and after periods of high precipitation or snowmelt.

oOo

DAMES & MOORE

CO-OP Mining Company
December 5, 1994

The following Plates are attached and complete this report.

Plate 1 - Fill Slope Cross Section TSF-1
Plate 2 - Fill Slope Cross Section TSF-2
Plate 3 - Fill Slope Cross Section TSF-3
Plate 4 - Fill Slope Cross Section TSF-4
Plate 5 - Fill Slope Cross Section TSF-5

Sincerely,

DAMES & MOORE, Inc.

LICENSED PROFESSIONAL ENGINEER
93-180048-2202
RUSSELL L.
OWENS
STATE OF UTAH

Russell L Owens

Russell L. Owens, P.E.
Senior Geotechnical Engineer
Professional Engineer, N. 180048
State of Utah

Curtis J. Tanner

Curtis J. Tanner, P.E.
Staff Engineer
Professional Engineer No. 184573
State of Utah

TSF-1

CO-OP Mine, Pad Section
Ten Most Critical. C:COOP1.PLT By: CJT 11-23-94 2:50pm

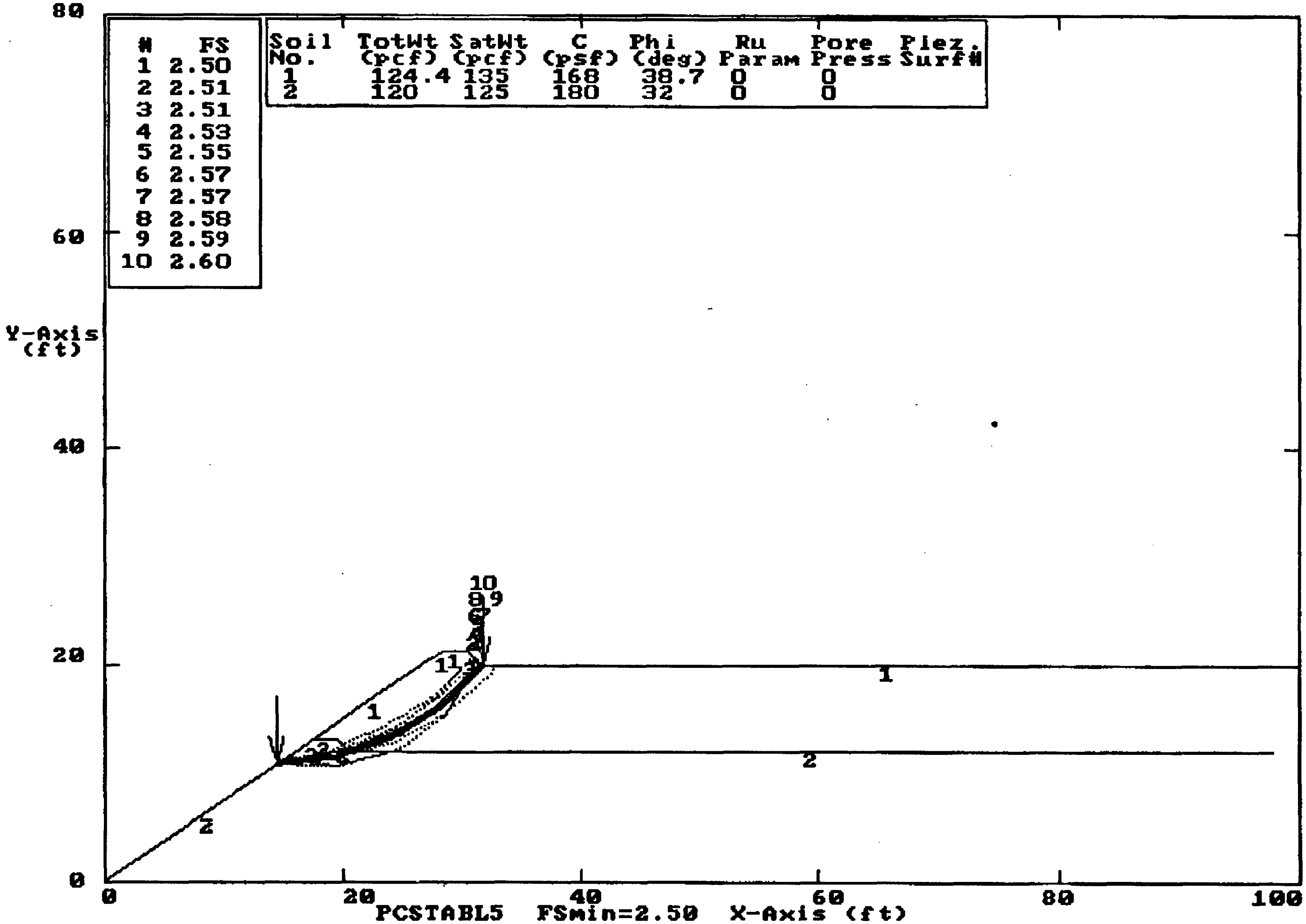
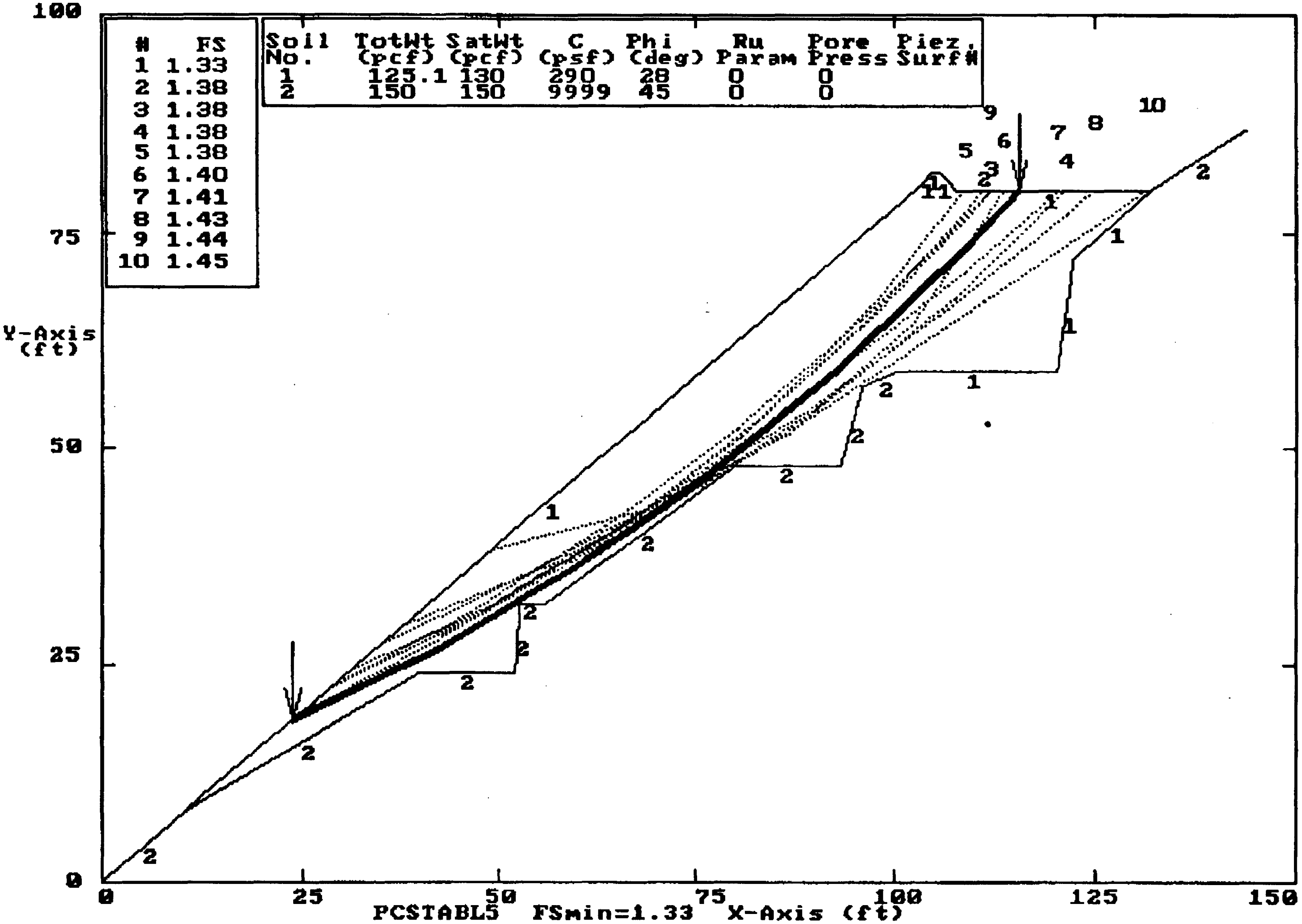


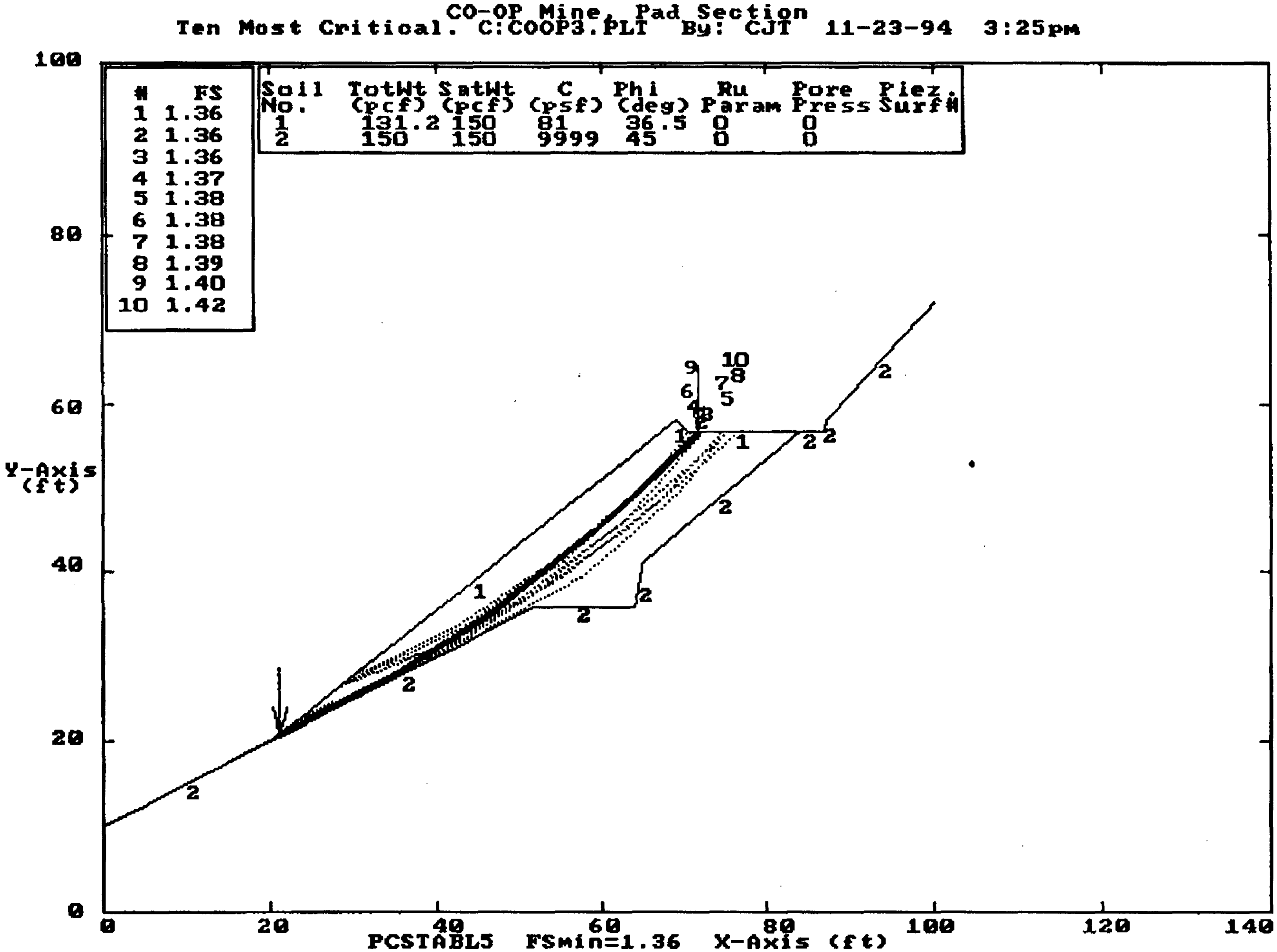
PLATE 1

TSF-2

CO-OP Mine, Pad Section
Ten Most Critical. C:COOP2.PLT By: CJT 11-23-94 2:57pm



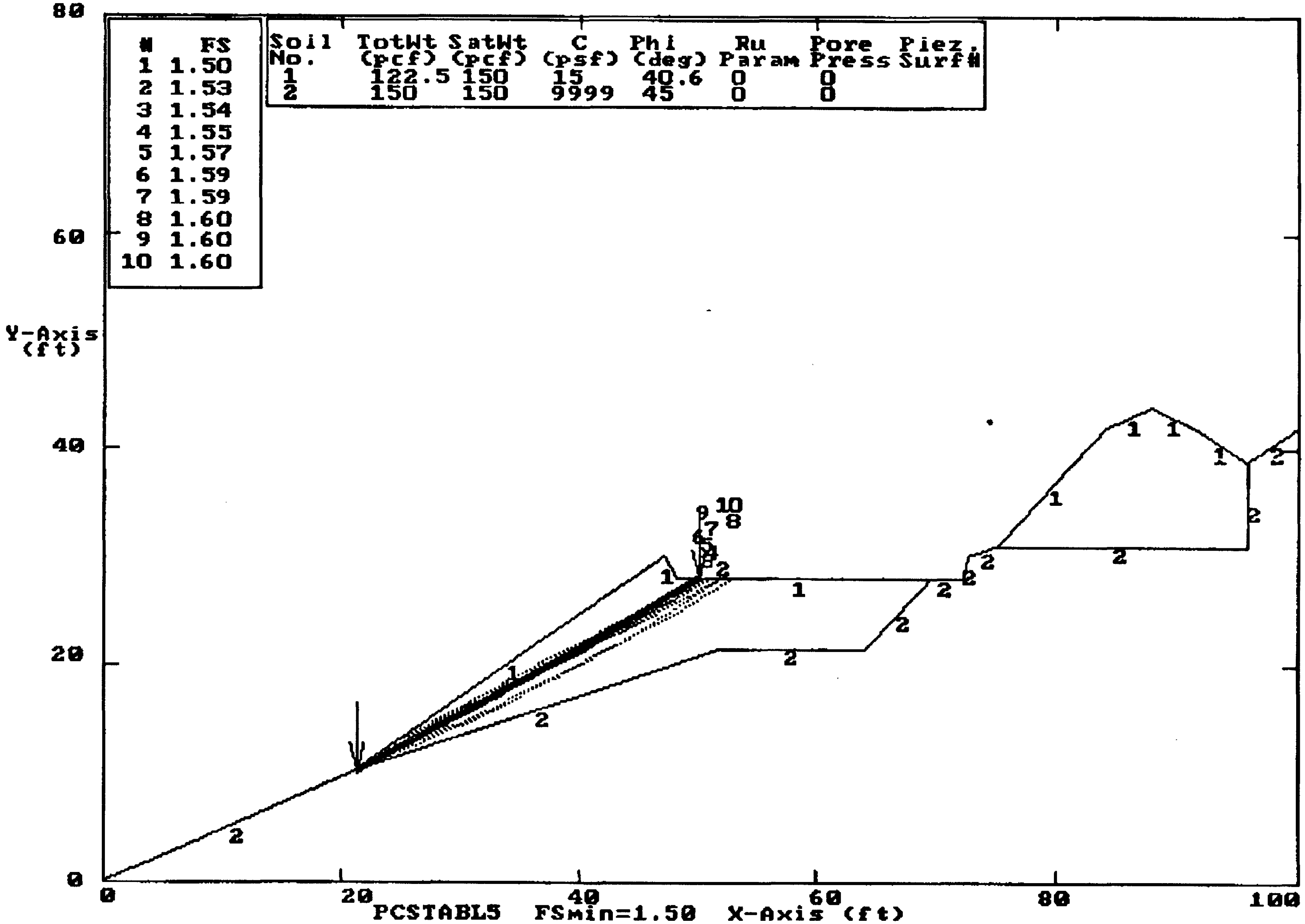
TSF-3



CO-OP Mine, Pad Section
Ten Most Critical. C:COOP3.PLT By: CJT 11-23-94 3:25pm
FS
1 1.36
2 1.36
3 1.36
4 1.37
5 1.38
6 1.38
7 1.38
8 1.39
9 1.40
10 1.42
Soil No. | TotWt (pcf) | SatWt (pcf) | C (psf) | Phi (deg) | Ru Param | Pore Press | Piez. Surf#
1 | 131.2 | 150 | 81 | 36.5 | 0 | 0 |
2 | 150 | 150 | 9999 | 45 | 0 | 0 |
Y-Axis (ft)
PCSTABL5 FSmin=1.36 X-Axis (ft)

TSF-4

CO-OP Mine, Pad Section
Ten Most Critical. C:COOP4.PLT By: CJT 11-23-94 3:19pm



TSF-5

CO-OP Mine, Pad Section
Ten Most Critical. C:COOP5.PLT By: CJT 11-23-94 3:22pm

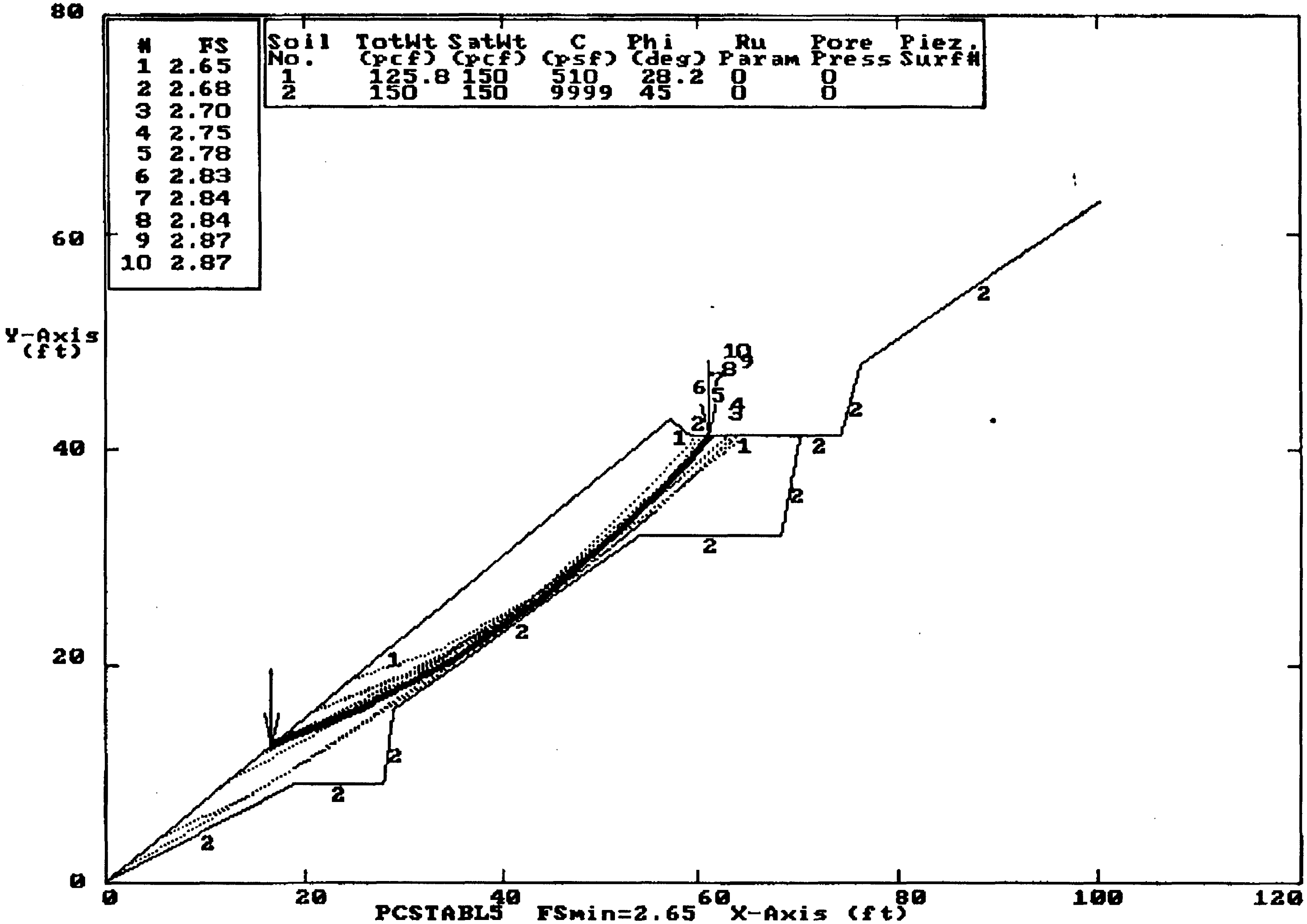
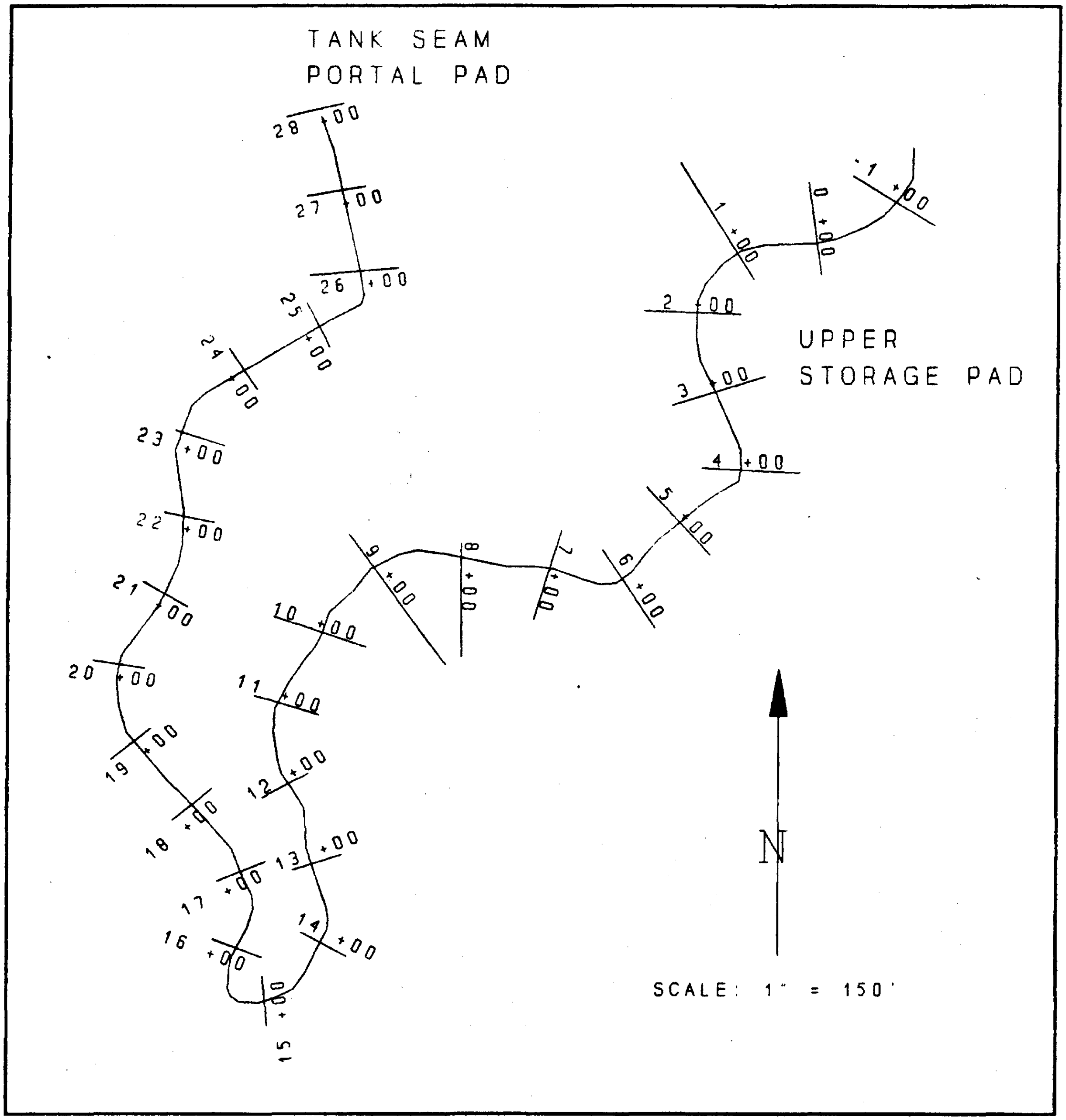


Table 5G-2 Tank Seam Road As-built Cut and Fill Summary

Station	Fill (-) Volume* (cu yd)	Cut (+) Volume* (cu yd)	Volume Cumulative (cu yd)	Station	Fill (-) Volume* (cu yd)	Cut (+) Volume* (cu yd)	Volume Cumulative (cu yd)
0+00			0	14+00			-9,576
	1,140	0			581	79	
1+00			-1,140	15+00			-10,078
	2,850	0			844	110	
2+00			-3,990	16+00			-10,812
	3,135	0			0	798	
3+00			-7,125	17+00			-10,014
	219	463			57	886	
4+00			-6,881	18+00			-9,185
	76	1,123			0	743	
5+00			-5,834	19+00			-8,442
	48	608			52	254	
6+00			-5,274	20+00			-8,240
	154	719			117	555	
7+00			-4,709	21+00			-7,802
	29	697			78	843	
8+00			-4,041	22+00			-7,037
	2,422	185			225	357	
9+00			-6,278	23+00			-6,905
	4,462	79			679	60	
10+00			-10,661	24+00			-7,524
	230	678			124	1,008	
11+00			-10,213	25+00			-6,640
	283	138			180	3,730	
12+00			-10,358	26+00			-3,090
	29	520			340	2,015	
13+00			-9,867	27+00			-1,415
	49	340			308	1,657	
14+00			-9,576	28+00			-66

*Cut and fill volumes measured using Quicksurf Version 5.1 3-D modeling software package, copywrite 1995, Schreiber Instruments, Inc., based on premining contours form 1991 aerial survey and contours shown on Plate 2-4E from 1995 aerial survey.

Figure 5G-5. Tank Seam As-built Road/Pad Stations Map



APPENDIX 5-H

SLOPE STABILITY ANALYSIS

Dames & Moore

250 East Broadway, Suite 200
Salt Lake City, Utah 84111
(801) 521-9255
TWX: 910-925-5692 Cable address: DAMEMORE

December 29, 1980

Mr. Wendell Owen
CO-OP Mining Company
Box 300
Huntington, Utah 84528

Dear Mr. Owen:

Report
Geotechnical Consultation
Bear Creek Portal
Near Huntington, Utah
For CO-OP Mining Company

INTRODUCTION

The purpose of this report is to discuss the stability of side-cast fill material placed during construction of the access road to the CO-OP Mining Company Bear Creek Portal. The scope of our consultation was formulated in discussions with Mr. Owen and consisted of a brief field reconnaissance, collection of a sample of the fill material, limited laboratory testing and preparation of this report.

The Bear Creek Portal is located in the SW 1/4, SW 1/4, S. 24, T. 16S., R. 7E., Emery County. Several abandoned facilities from a previous mining effort exist near the Portal. We understand that the CO-OP Mining Company is in the process of re-opening the old mine and that the existing old Portal will be used for ventilation of the new mine. The mine is located in a steep slope in the Wasatch Plateau; access to the Portal is by a typical unsurfaced access road constructed by conventional side-cast methods.

We understand that a violation of the federal regulations (Section 717.14 in the Federal Register, Vol.42, No.239, P.62695, issued December 13, 1977) was issued by the State Division of Oil Gas and Mining to CO-OP Mining Company. Part (c) of this section states that surface operations on steep slopes (20 degrees or more) "shall be conducted so as not to place any material on the down slope below road cuts, mine workings or other benches, other than in conformance with part (a)(1)" of Section 717.14. Part (a)(1) states that fills shall achieve a minimum static safety factor of 1.5.

The slope where the Bear Creek Portal is located is steeper than 20 degrees and the only feasible way to construct the access road is by conventional side-cast methods. Therefore, the static factor of safety of the side-cast material is the key issue.

DISCUSSION

The material being excavated and forming the side-cast fill is gravel and cobble sized pieces of silty sandstone in a sandy and silty clay matrix. Calcium carbonate derived from the cement in the sandstone is also present. The results of a partial grain size analysis are tabulated below:

SIEVE NUMBER	PERCENT FINER BY WEIGHT
#4	72.2
#40	64.0
#200	37.2

The results of an Atterberg Limit determination performed on the material finer than the number 200 sieve are tabulated below:

LIQUID LIMIT(%)	PLASTIC LIMIT (%)	PLASTICITY INDEX	UNIFIED SOIL CLASSIFICATION
30.5	15.6	14.9	CL

During our reconnaissance, we examined the access road leading to the Trail Canyon Portal in Section 22. This road is in virtually indentical conditions to the road leading to the Bear Creek Portal. Some of the side-cast fill near the Trail Canyon Portal is 25 years old; the youngest of it is about 10 years old. The side-cast fill material near the Trail Canyon Portal is nearly everywhere sloping at 35 degrees (approximately 1-1/2 horizontal to 1 vertical).

The side-cast material appears to be performing in a satisfactory manner. In many places, the side-cast material appears to be very similar to natural slopes between nearly vertical exposures of resistant sandstone. Very minor gullies and slumps are present locally. Vegetation is becoming established on some of the slopes.

The surface of the side-cast material is quite firm and difficult to walk on because boot heels don't penetrate. We believe that the reason the surface is so firm is related to the clay and the calcium carbonate in the soil. The clay gives the soil cohesive strength and the calcium carbonate tends to cement the soil particles together.

The calcium carbonate cement in the soil is probably a significant constituent in the safety factor of side-cast fill material which has remained stable for 25 years in a situation identical to the access road leading to the Bear Creek Portal.

CONCLUSION

Based on observations made during our reconnaissance and the discussions presented above, it is our opinion that the side-cast material located adjacent to the access road leading to the Bear Creek Portal will behave in a manner similar to the side-cast material located adjacent to the access road leading to the Trail Canyon Portal. Since the material near the Trail Canyon Portal has been grossly stable for 10 to 25 years as determined by its performance, we believe that the material placed during construction of the Bear Creek Portal access road will also be grossly stable.

Consequently, we believe that the stability concern identified in the regulations can best be addressed by the empirical evidence of the performance of similar material in a similar situation rather than a calculated factor of safety.

We trust that this report satisfies your present needs. If you have any questions or require additional discussions or information, please contact us.

Very truly yours,

DAMES & MOORE

William J. Gordon
Associate
Professional Engineer No. 3457
State of Utah

Jeffrey R. Keaton
Engineering Geologist

WJG/jrk/kc

Dames & Moore
250 East Broadway, Suite 200
Salt Lake City, Utah 84111
(801) 521-9255
TWX: 910-925-5692 Cable address: DAMEMORE

February 20, 1981

Mr. Wendell Owen
Co-op Mining Company
Box 300
Huntington, Utah 84528

Dear Mr. Owen:

Summary Report
Slope Stability Analyses
Bear Creek Portal
Access Road
Near Huntington, Utah
For Co-op Mining Company

INTRODUCTION

This report summarizes the results of our stability analyses of the slopes along the Bear Creek Portal Access Road located northwest of Huntington, Utah.

PURPOSE AND SCOPE

The purpose and scope of this study were planned in discussions between Mr. Wendell Owen of Co-op Mining and Mr. Bill Gordon of Dames & Moore. In general, the purpose of this investigation was to analyze the static factor of safety of the side-cast cut and fill slopes along the Bear Creek Portal Access Road.

BACKGROUND

The Co-op Mining Company is in the process of reopening an abandoned coal mine at the Bear Creek Portal. Several abandoned facilities from a previous mining effort exist near the portal. We understand that the existing old portal will be used for ventilation of the new mine. The mine is located on a steep slope in the Wasatch Plateau and access to the portal is by a typical unsurfaced access road constructed by conventional side-cast methods.

Co-op Mining Company was issued a citation by the Department of Natural Resources Division of Oil, Gas, and Mining. The nature of the violation was with regard to the placement of side-cast cut and fill material on steep slopes (20 degrees or more). Regulations require that such fills achieve a minimum static factor of safety of 1.5.

An engineering geologist from Dames & Moore previously visited the site and performed a reconnaissance survey of the area and sideslopes in question. Laboratory tests have been performed on samples of the side-cast cut and fill material obtained at the site. These laboratory tests included sieve analyses and Atterberg Limits. The results of these laboratory tests, a discussion of our site reconnaissance survey, and a summary of our conclusions were presented in a report dated December 29, 1980*.

*"Report, Geotechnical Consultation, Bear Creek Portal, Near Huntington, Utah, For Co-op Mining Company."

SITE CONDITIONS

The general location of the Bear Creek Portal Access Road is shown on Plate 1, Plot Plan. Side-cast cut and fill areas as determined by others are also indicated on Plate 1. The slopes in the area of the Bear Creek Portal are generally steeper than 20 degrees and the access road has been constructed by conventional side-cast methods. The material being excavated and forming this side-cast cut and fill typically consists of fine and coarse gravel and cobble sized pieces of silty sandstone in a sandy and silty clay matrix. Calcium carbonate derived from the cement in the sandstone is also present.

The surface of the side-cast material is quite firm, which we believe to be related to the composition of clay and calcium carbonate in the soil. The clay acts as a binder and gives the soil cohesive strength and the calcium carbonate tends to cement the soil particles together. As discussed in our previous letter, the calcium carbonate cement in the soil probably provides a significant component of the factor of safety of the side-cast fill material. However, the determination of a numerical value for the influence of the calcium carbonate cementation would be very difficult to accurately determine.

SOIL PROPERTIES

Based on the results of laboratory tests performed on samples of the side-cast cut and fill material from the Bear Creek Portal site and our experience with similar soils, we have assumed the following soil properties:

Side-Cast Fill Material

Angle of Internal Friction	$\phi = 26^{\circ}$
Cohesion	C = 350 psf
Unit weight soil	δ = 98 pcf

Natural Soils

Angle of Internal Friction	$\phi = 26^{\circ}$
Cohesion	C = 700 psf
Unit weight soil	δ = 120 pcf

SLOPE STABILITY ANALYSIS

To aid in evaluating the stability of the side-cast cut and fill material of the Bear Creek Portal Access Road, a computer slope stability analysis was performed. The computer analysis utilized a simplified Bishop's Method in computing the long-term static factor of safety of the slopes. Due to the limited laboratory and field data, and the uncontrolled method in which side-cast cut and fill materials are placed, ultra conservative soil strength parameters were used in the computer analysis. A Geometric cross-section of a critical section utilized in the analysis is shown on Plate 2, Slope Cross Section. It was also assumed that a phreatic water surface would not develop in the slopes of the embankment.

The computer program analyzed the slope stability by searching a specified coordinate grid area for the center of the circle having the lowest factor of safety. The slope stability analyses was performed using a total of four separate coordinate grid areas. The number of trial failure arc centers analyzed in each of these four areas varied from 12 to 63. As indicated on Plate 2, this analysis indicated a minimum static factor of safety varying from 1.43 to 2.15.

Copies of the results of the computer analysis for each coordinate grid area are included with this report.

DISCUSSIONS AND RECOMMENDATIONS

GENERAL

Supporting data upon which our recommendations are based have been presented in the previous sections of this report and in the previous Dames & Moore report dated December 29, 1980.

SLOPE STABILITY

The computer slope stability analysis indicates a minimum static factor of safety varying from 1.43 to 2.15 for the trial arcs analyzed.

It should be noted that the factor of safety of the trial arc which cuts deep into the slope does not consider the presence of bedrock, increasing strength of the natural soils with depth, or the effect of the calcium carbonate cementation in the soil. If the above were incorporated into the analysis, the factor of safety would be significantly higher.

Stability of the slopes will be influenced by the degree of saturation of the existing soils. Therefore, surface drainage must be channeled to minimize runoff over the slopes. However, during wet periods of the year, small localized slides and sloughs should be anticipated along the slopes. However, these occurrences should be minor. The performance of these side-cast cut and fill slopes is anticipated to be similar to virtually identical side-cast cut and fill slopes along the nearby road leading to the Trail Canyon Portal. These slopes have been stable since their construction, varying from 10 to 25 years ago.

Based on our slope stability analysis and observations made during our reconnaissance visit to the site, it is our opinion that the side-cast fill material located along the Bear Creek Portal Access Road generally has a long-term static factor of safety of 1.5 or greater and will perform in a satisfactory manner.

o0o

We appreciate the opportuniqy of performing this service for you. If you have any questions or require additional information, please contact us.

Very truly yours,

DAMES & MOORE

William J. Gordon
Associate
Professional Engineer No. 3457
State of Utah

Douglas G. Beck

Douglas G. Beck
Staff Engineer

WJG/DGB/wb

Attachments"

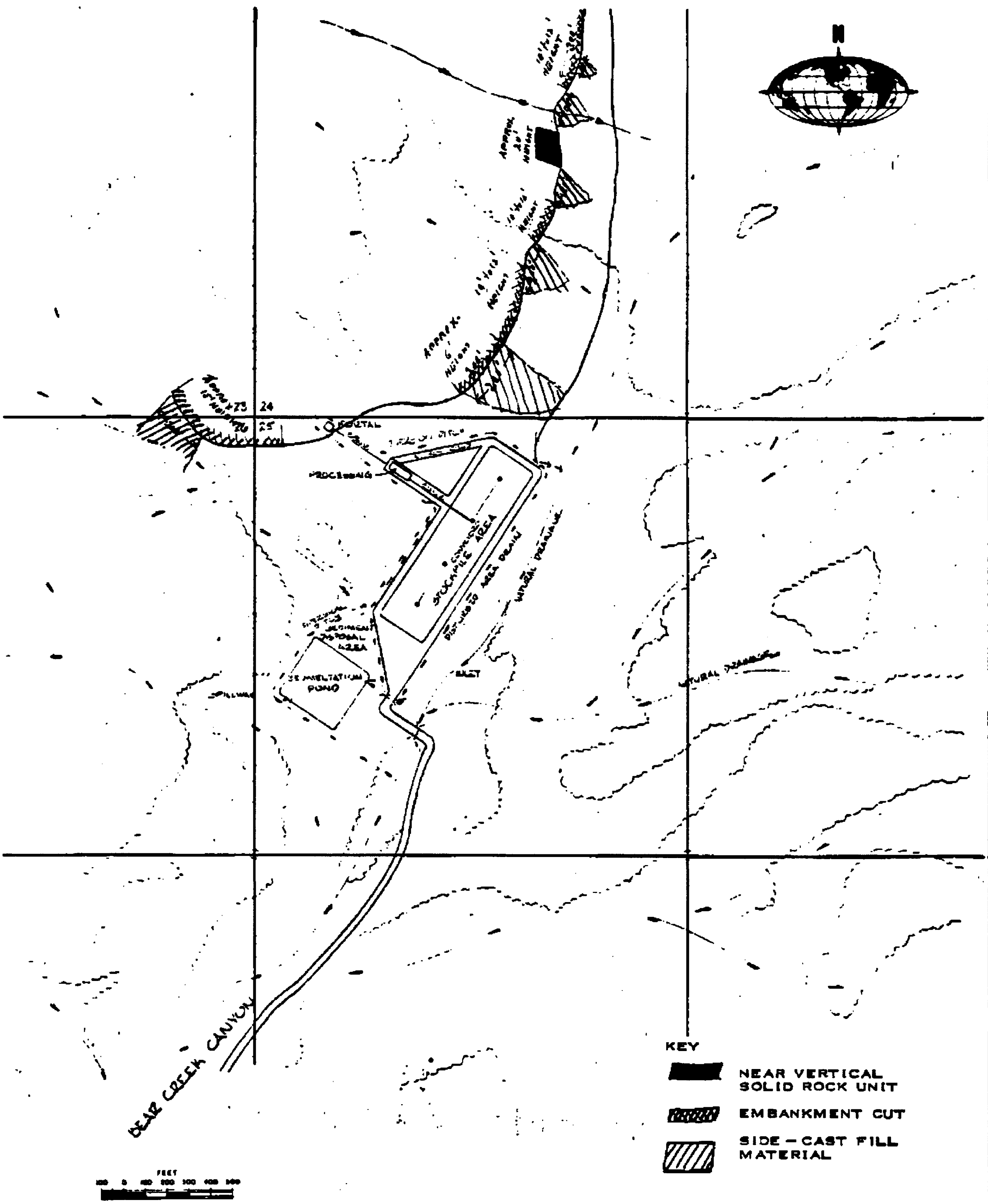
Plate 1 - Plot Plan
Plate 2 - Slope Cross-Section
Computer Anlaysis Results

cc: Department of Natural Resources
Division of Oil, Gas and Mining (2)

REVISIONS
BY ________ DATE ________

FILE ________

BY ________ DATE ________
CHECKED BY ________

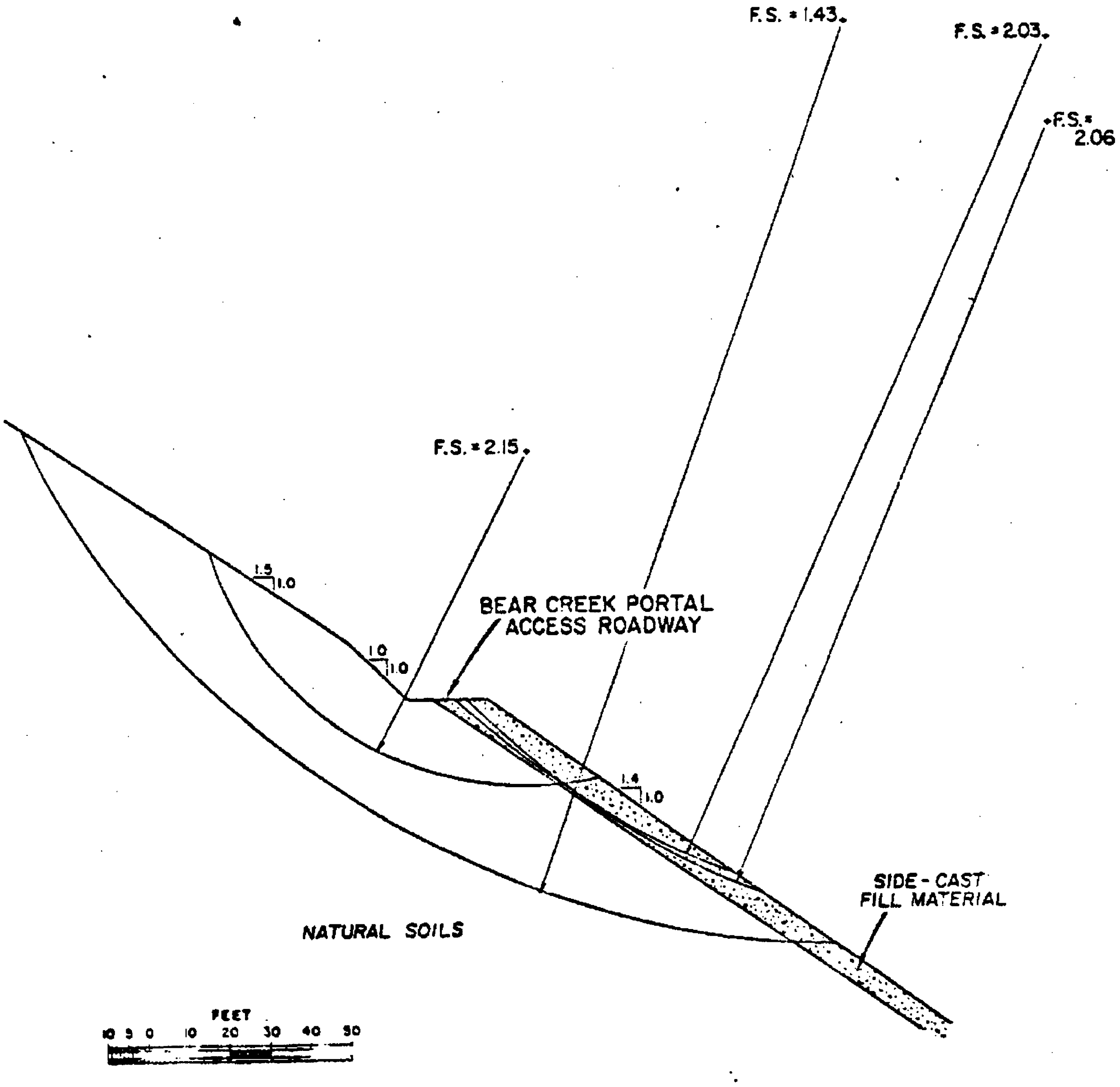


PLOT PLAN

REFERENCE
ADAPTED FROM PRINT
SUPPLIED BY OTHERS.

DAMES & MOORE

PLATE 1



SLOPE CROSS SECTION

SLOPE STABILITY ANALYSIS - SIMPLIFIED BISHOP'S METHOD

DATE OF LAST REVISION - DEC 11 78

DATE RUN - 02/19/81 TIME RUN - 19.00.45

0546101206 6093 GMB 022081 STABILITY ANALYSIS CUTSLOPE PARTIAL FILL

7

****

DATA INPUT MODE = 1
EARTHQUAKE COEFFICIENT = 0.000
PORE PRESSURE IS DEFINED BY WATER LINE DATA
TOTAL NUMBER OF SOIL LINES = 6 NUMBER OF WATER LINES = 0
0SLOPE GEOMETRY DATA

LINE NO	LEFT-X	LEFT-Y	RIGHT-X	RIGHT-Y	SOIL DATA- WT BELOW LINE-PCF	FRICT.ANGLE(DEG) (OR C/P-RATIO**) ABOVE	FRICT.ANGLE BELOW	COHESION-PSF ABOVE	COHESION-PSF BELOW	PORE PRESSURE RATIO ABOVE	PORE PRESSURE RATIO BELOW	NEW BELOW	NUU ABOVE	NUU BELOW
1	1018.00	1150.00	1086.00	1134.00	120.0	0.00	26.00	0.0	700.0	0.000	0.000	0	0	0
2	1086.00	1134.00	1100.00	1120.00	120.0	0.00	26.00	0.0	700.0	0.000	0.000	0	0	0
3	1100.00	1120.00	1106.00	1120.00	120.0	0.00	26.00	0.0	700.0	0.000	0.000	0	0	0
4	1106.00	1120.00	1120.00	1120.00	98.0	0.00	26.00	0.0	350.0	0.000	0.000	0	0	0
5	1120.00	1120.00	1232.00	1040.00	98.0	0.00	26.00	0.0	350.0	0.000	0.000	0	0	0
6	1106.00	1120.00	1220.00	1042.00	120.0	26.00	26.00	350.0	700.0	0.000	0.000	0	0	0

NOTE: IF(NEW.EQ.1) SOIL IS NEWLY PLACED AND DOES NOT CONSOLIDATE LAYERS WITH NUU=1
IF(NUU.EQ.1) SOIL WILL BE LOADED UNDER UNDRAINED CONDITIONS BY NEWLY PLACED LAYERS
VALUES MARKED WITH ** ARE C/P RATIOS FOR LAYERS WITH NUU=1

UNIT WEIGHT OF WATER = 62.40 NUMBER OF COLUMN LOADS = 0

MODE OF PROGRAM OPERATION = 3

CENTER VARIATIONS	MIN-X	MAX-X	MIN-Y	MAX-Y	DX	DY
	1240.00	1260.00	1250.00	1280.00	10.00	10.00

RADIUS TANGENTS MAX= 1058.00 MIN= 1062.00 INCR= 1.00

1

RESULTS

****

ENTR NO	RAD	CENTER COORDINATES	CIRCLE	FACTOR OF SAFETY	SUMWS	SUM1	SUM2	XR	XL	ARC	NN	TRIAL
	N060.26	170.08	2									

TRM
RDY
EDIT,OLD,GMB
UNIEDIT 3.32 READY ? WIDTH=/132/*FIND:/TABUL/*P.1;*
1

TABULATION OF MINIMUM SAFETY FACTORS
(CRITICAL RADIUS IN PARENTHESES)

Y COORDINATES \ X COORDINATES	1240.0	1250.0	1260.0
1280.0	1.802 (222.)	2.111 (222.)	2.026 (219.)
1270.0	1.877 (212.)	2.034 (212.)	2.108 (210.)
1260.0	1.953 (202.)	2.114 (202.)	2.063 (202.)
1250.0	1.954 (191.)	2.084 (188.)	2.575 (192.)

READY ? END
RDY-FOR
BYE
CT = 00.23 SU-A = 1.9
KCM = 7
D034008 LOG OFF. 19.04.00.

2/*FIND:/TABUL/*P.1;*
TABULATION OF MINIMUM SAFETY FACTORS
(CRITICAL RADIUS IN PARENTHESES)

Y COORDINATES \ X COORDINATES	1210.0	1215.0	1220.0	1225.0	1230.0	1235.0	1240.0
1310.0	1.534 (240.)	1.504 (244.)	1.478 (248.)	1.451 (252.)	1.495 (252.)	1.549 (252.)	1.617 (252.)
1305.0	1.512 (237.)	1.486 (241.)	1.431 (247.)	1.467 (247.)	1.515 (247.)	1.569 (247.)	1.646 (247.)
1300.0	1.462 (236.)	1.440 (240.)	1.445 (242.)	1.485 (242.)	1.531 (242.)	1.594 (242.)	1.672 (242.)
1295.0	1.449 (233.)	1.427 (237.)	1.460 (237.)	1.501 (237.)	1.552 (237.)	1.619 (237.)	1.701 (237.)
1290.0	1.437 (230.)	1.441 (232.)	1.477 (232.)	1.518 (232.)	1.575 (232.)	1.643 (232.)	1.734 (232.)
1285.0	1.426 (227.)	1.456 (227.)	1.491 (227.)	1.538 (227.)	1.595 (227.)	1.670 (227.)	1.766 (227.)
1280.0	1.440 (222.)	1.471 (222.)	1.509 (222.)	1.559 (222.)	1.618 (222.)	1.699 (222.)	1.802 (222.)
1275.0	1.455 (217.)	1.485 (217.)	1.528 (217.)	1.577 (217.)	1.645 (217.)	1.727 (217.)	1.836 (217.)
1270.0	1.468 (212.)	1.503 (212.)	1.545 (212.)	1.600 (212.)	1.669 (212.)	1.759 (212.)	1.877 (212.)

1

TABULATION OF MINIMUM SAFETY FACTORS
(CRITICAL RADIUS IN PARENTHESES)

Y COORDINATES \ X COORDINATES	1140.0	1145.0	1150.0	1155.0	1160.0
1180.0	2.425 (82.)	2.675 (82.)	3.063 (82.)	3.269 (81.)	2.729 (80.)
1175.0	2.536 (77.)	2.819 (77.)	3.262 (77.)	2.883 (73.)	2.744 (75.)
1170.0	2.662 (72.)	2.982 (72.)	3.294 (70.)	2.867 (68.)	2.694 (71.)
1165.0	2.811 (67.)	3.167 (67.)	3.041 (67.)	2.703 (64.)	2.479 (67.)
1160.0	2.984 (62.)	3.235 (60.)	2.997 (62.)	2.563 (60.)	2.519 (62.)

TABULATION OF MINIMUM SAFETY FACTORS
(CRITICAL RADIUS IN PARENTHESES)

Y COORDINATES \ X COORDINATES	1120.0	1125.0	1130.0	1135.0	1140.0
1180.0	5.342 (82.)	2.219 (78.)	2.146 (82.)	2.260 (82.)	2.425 (82.)
1175.0	5.061 (77.)	2.173 (76.)	2.225 (77.)	2.352 (77.)	2.536 (77.)
1170.0	2.181 (71.)	2.220 (72.)	2.315 (7[illegible].)	2.456 (72.)	2.662 (72.)
1165.0	2.241 (67.)	2.314 (57.)	2.421 (67.)	2.577 (67.)	2.811 (67.)

Appendix 3-F (cont)

5/10/85

STABILITY ANALYSIS

HIGHWALL STABILITY

Highwalls at the site face south, southeast, and east. The highwalls are nearly vertical, with an average slope of IV:2H, or 80.

The nighwall stability analysis is based on a rotational shear analysis using the Hoek method. Compressive strengths of materials in the Blackhawk Formation are highly variable, ranging from 290 psi for soft shale to more than 20,000 psi for certain sandstones, An average value of 5,000 psi has been used for this analysis, This is a very conservative figure, based on the relative proportions of sandstones and shales in the exposed highwalls.

There are 2 joint sets typically found in this area. the major set has a strike of about N 10 deg E and dips 80 deg to vertical. The minor set has a strike of approximately N 70 deg W and also dips greater than 80 deg. The bedding in the highwall area is nearly flat.

Cohesion can be calculated from compressive strength by the following formula:

$$C_i = \frac{Co}{2}\tan\left(45-\frac{\theta}{2}\right);$$

where: C_i = Intact rock shear strength or cohesion

Co = Intact rock compressive strength

θ = Internal friction angle.

Using a typical internal friction angle of 45 deg for Wasatch Plateau rock types, and a 5,000 psi compressive strength, a cohesion or intact rock strength of approximately 1,000 psi is found. Since the 1,000 psi value is for intact or solid rock, the value must be adjusted to compensate for jointing and fracturing common to all rock masses. A method of relating fracture intensity and cohesion was developed by Stimpson and Ross-Brown and can be found in the article entitled, "Estimating the Cohesive Strength of Randomly Jointed Rock Masses", Mining Engineering, Vol. 31, No. 2, pp. 182-188. Based on this method and using a conservative figure of 4 joints per meter, a .065 factor is determined for calculating rock mass cohesion (C_m). Based on a C_i of 1,000 psi, C_m becomes 65 psi.

A typical or average rock mass bulk density of 155 lbs./ft^3 was selected for the analysis, and a slightly conservative, but commonly used value of 31 deg was selected for the rock mass sliding friction angle.

The following parameters were used with the Hoek slope chart (Hoek, E., and J.W. Bray, 1981, Rock Slope Engineering, Revised Third Edition, IMM, London):

H = Maximum Slope Height - 100 ft

θ = Slope Angle (average) - 80 deg

C_m = Rock Mass Cohesion - 65 psi

ϕ = Rock Mass Friction Angle - 31 deg

γ = Rock Mass Bulk Density - 155 lbs./ft^3

Plotting the above parameters on the Circular Failure Charts Nos. 1 and 5 (Figure 3F-1 and 3F-2), it can be seen that the projected highwalls will have a safety factor or 2.61 under dry conditions and 2.40 under saturated conditions. It should be noted that the safety factors exceed the required 1.5 safety factor.

EMBANKMENT STABILITY

Embankment or backfill will be placed in lifts not to exceed 36 in. and will be compacted to 90 pct of the laboratory obtained T-99 Standard Proctor. Slopes will not exceed IV:1.5H or 33.7 deg. Soil properties are based on those used in the "Slope Stability Analyses for the Bear Creek Portal and Access Road", by Dames and Moore, February 20, 1981, and the "Geotechnical Consultation, Bear Creek Portal", by Dames and Moore, December 29, 1980, and the "Bear Canyon Mine Site, Sedimentation Pond "A" Stability Analysis", by

Horrocks and Corollo Engineers on July 12,1984.

Based on the proposed plan, and the results of samples taken during the above studies, the following parameters were established for the safety factor calculations:

a. H = Embankment Height = 30 ft; this represents the maximum height of compacted embankment proposed in the plan;

b. θ = Slope Angle = 33.7 deg; this is the maximum slope (IV:1.5H) proposed for the reclaimed embankments.

c. C_m = Soil Cohesion @ 90 pct Compaction = 4.375 psi; Actual Cohesion tests on compacted native material at this site showed a cohesion value of 700 psf at a density of 118 lbs./ft^3 and a compaction value ranging from 89 pct to 94 pct. (See Sediment Pond "A" Stability Analysis, by Horrocks & Corollo Engineers, July 12, 1984.) To provide for maximum safety in the calculation, the cohesion factor is reduced by the compaction factor.

700 psf X 0.9 = 630 psf = 4.375 psi

d. ϕ = Friction Angle = 26 deg; This angle is based on the measurements taken and reported in the Feb. 20, 1981 Dames & Moore Slope Stability Analysis on the Bear Creek

Portal and Access Road.

e. Y = Rock Mass Bulk Density (90 pct) = 108 lbs./ft^3; Once again, this is a conservative number, established by taking actual values of 118 to 120 lbs./ft^3 as reported in the above reference stability analysis, and allowing for 90 pct compaction - 120 X 0.90 = 108 lbs./ft^3.

A rotational shear analysis was performed using the Hoek method to determine stability of the backfilled slopes. The following parameters were used for the slopes:

H = Embankment Height - 30

θ = Slope Angle - 33.7

C_m = Soil Cohesion @ 90 pct Compaction - 4.375 psi

ϕ = Friction Angle - 2

Y = Rock Mass Bulk Density (90 pct) - 108 lbs./ft^3

Based on the above criteria, backfilled slopes are found to have an expected safety factor of a maximum of 2.21 for dry conditions to a minimum of 1.68 for saturated conditions. Both cases exceed the required static safety factor of 1.3. It should also be noted that the previous slope stability analysis by Dames & Moore resulted in static safety factors ranging from a minimum of 1.43 to 2.15 for side-cast cut and fill material in this area.

Note: The embankment compaction factors and cohesion values are based on previous tests performed in the Bear Canyon area. Although the tests were not site specific, they were run on the existing soils which are the same as those to be used in reclamation. The values used for rock compressive strengths were taken from rock parameters typical of the Blackhawk Formation in the Wasatch Plateau, and commonly used and accepted for this type of calculations.

If it is determined necessary, Co-Op will commit to taking site specific tests on the soils and highwall rock to further verify the factors of safety, these tests would be performed prior to reclamation, at the discretion of the Division.

REMOVAL OR REDUCTION OF HIGHWALLS

Due to the laws requiring removal of highwalls constructed following 1977, the Division has directed Co-Op to recover all highwalls. Plates 3-2 show existing highwalls (Plates 2-4) are recovered during reclamation.

Figure 3F-1
(DRY CONDITIONS)

CIRCULAR FAILURE CHART NUMBER 1

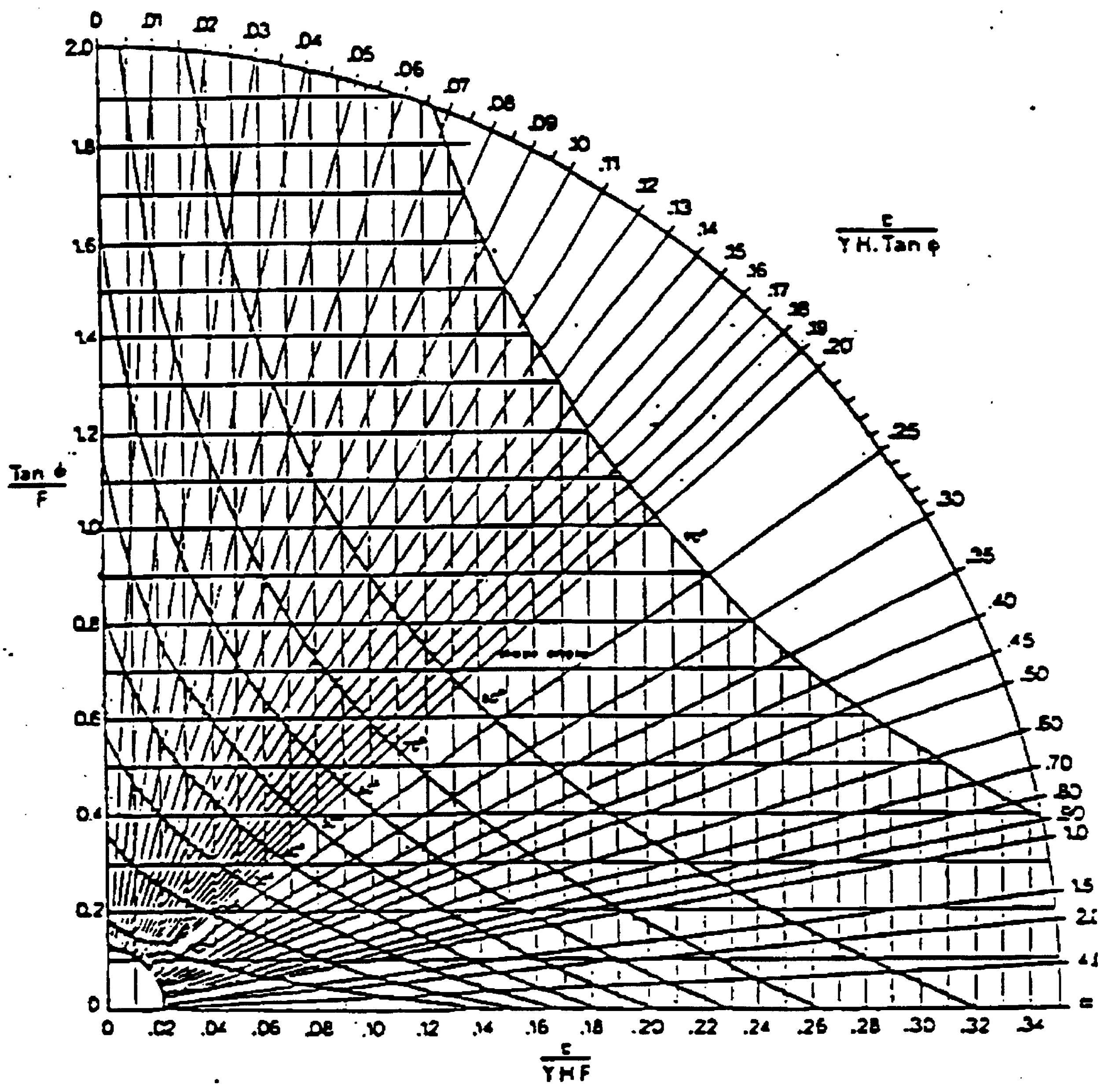
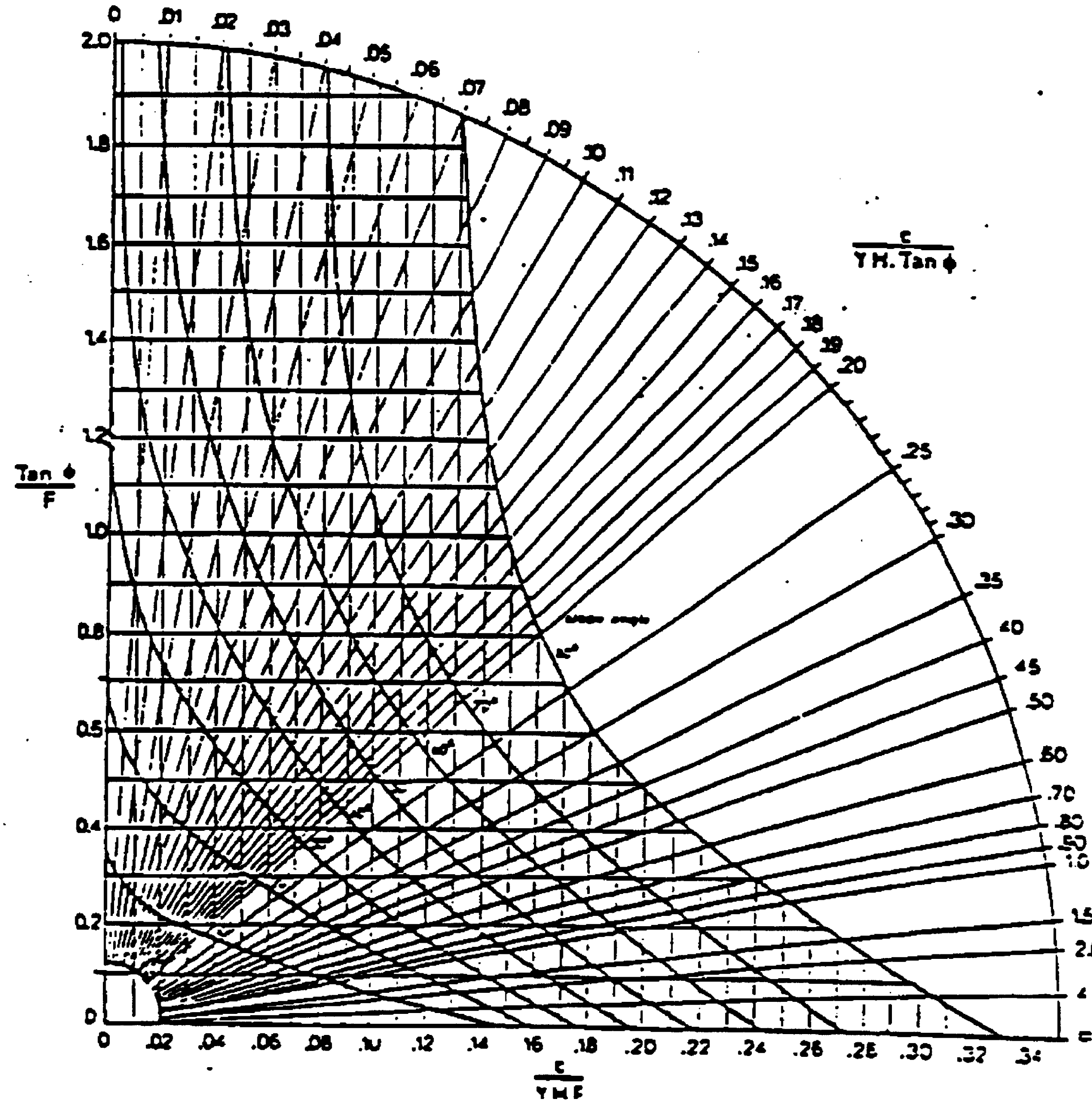


Figure 3F-2

(SATURATED CONDITIONS)

CIRCULAR FAILURE CHART NUMBER 5



BY D. Guy DATE 8/15/85 SUBJECT Highwall Lithologic Section SHEET NO. .... OF ....

CHKD. BY .... DATE .... JOB NO. ....

Scale: 1"=20'

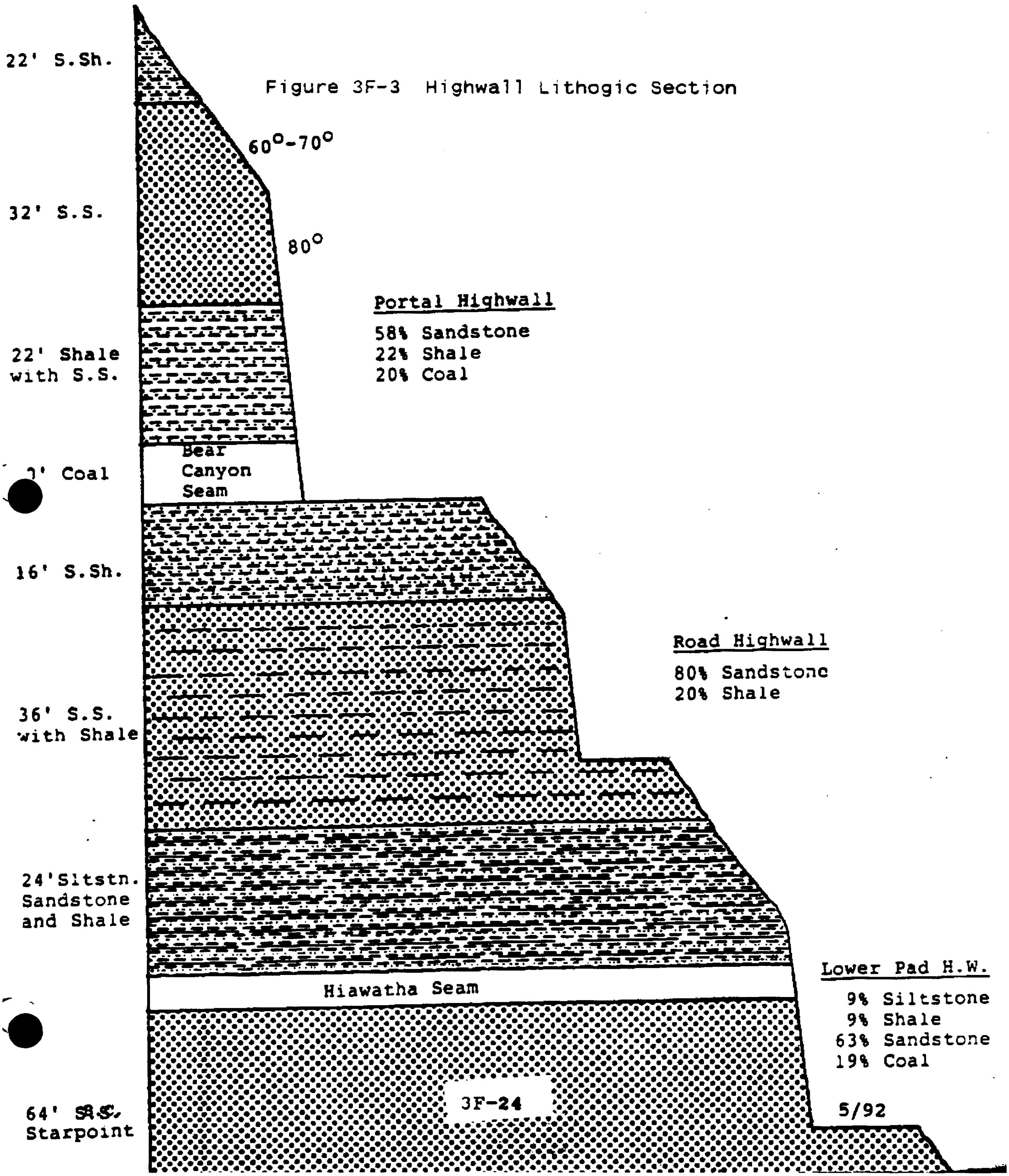


Figure 3F-3 Highwall Lithogic Section

RECLAIMED TANK SEAM PORTAL ACCESS ROAD

SLOPE STABILITY ANALYSIS

DAMES & MOORE

127 SOUTH 500 EAST, SUITE 300, SALT LAKE CITY, UTAH 84102-1959
(801) 521-9255 FAX: (801) 521-0380

May 10, 1994

CO-OP Mining Company
P.O. Box 1245
Highway 31
Huntington Canyon
Huntington, Utah 84528

Attention: Mr. Charles Reynolds
Mining Engineer

Report
Stability Analysis for Reclaimed
Tank Seam Portal Access Road
Job No. 27437-001-162

INTRODUCTION

Dames & Moore performed stability analyses for construction and reclamation of the Tank Seam Portal Access Road in 1993. Minor revisions were incorporated into that report, dated September 16, 1993 for access road construction, and an updated report has been issued. This report incorporates updated reclamation concepts and replaces the original stability analysis for the reclaimed Tank Seam Portal Access Road, dated September 22, 1993.

STABILITY ANALYSIS

This report presents stability analyses for the anticipated configuration of the Reclaimed Tank Seam Portal Access Road. The general location of the proposed access road is presented on Plate 1. It is assumed that the access road excavation will be reclaimed to approximately the same slope as the topography adjacent to the access road alignment. Backfill materials will be derived from the fill sections of the road and from adjacent slopes during the reshaping. Materials to be used during reclamation of the access road will be native soils and no additional materials from offsite borrow sources are anticipated. Boulders obtained during initial road construction that are not incorporated into reclamation fill and that cannot be broken down into smaller fragments will be placed on stable flat areas of the mine site.

A stability analysis of the proposed reclaimed slope was performed utilizing a two dimensional, limit equilibrium stability program called PCSTABL5. An automatic search routine was employed to determine the failure surfaces with the lowest factors of safety. Only circular failure surfaces were evaluated. A natural slope of 35 degrees was modeled considering dry conditions.

The analysis of the cut slope section considered one horizontal to two vertical (1H:2V) cuts. Bedrock was modeled to be present 6 feet vertically below the ground surface and to trend parallel to the natural cut. The roadway section cut was backfilled to conform to the natural existing slopes above and below the access road excavation. The small fill section, which includes the safety berm, was left in place in the stability analysis model. However, it could be removed during reclamation activities. Removal of the berm would not decrease the stability of the slope. Plate 2 presents the configuration modeled, the input material properties, and the ten failure surfaces with the minimum safety factors. A minimum factor of safety of 1.8 was determined for the input configuration. The "arrows" on Plate 2 indicate the failure surface with the lowest safety factor.

The fill material placed in the road cuts should be compacted in lifts not exceeding 18 inches in depth and compacted by heavy machinery during placement. All rock fragments in excess of 18 inches should be removed from the initial lifts of the fill. Rock fragments obtained during road construction and incorporated in the fill should be placed in a manner to minimize void space. During placement of the fill, boulder sized rock fragments in excess of 18 inches could be incorporated into upper lifts of the fill provided the majority of the fragments are well embedded in the fill and the material adjacent to these rock fragments is properly compacted. The surface of the reclaimed ground that is devoid of protruding rock fragments should be scarified to a minimum depth of 12 inches to reduce surface compaction and to aid in the re-establishment of vegetation.

All of the existing gullies for slope drainage should be re-established to their natural configuration. Since the stability of the reclaimed slopes will be influenced by the degree of saturation of the existing soils, surface drainage must be channeled to prevent or at least minimize runoff over the reclaimed slopes.

Questions have been raised concerning the stability of the scarified surface of the reclaimed slopes, particularly if topsoil comprises the majority of the scarified material. To evaluate this scenario, a reclaimed slope of 35 degrees with simulated 18 inch lifts was modelled. The friction angle of the outer 12 inches of the slope was reduced to 30 degrees to mimic scarified conditions. We understand that an erosion control/revegetation blanket similar to Excelsior S-2 or equivalent will be placed over the scarified slope surface and stapled into scarified material to minimize erosion potential and to enhance slope stability. This blanket was also incorporated into the model and a minimal tensile strength utilized. The modelled configuration, input properties, and the ten failure surfaces with the minimum factors of safety are presented on Plate 3.

The analysis indicates that the scarified material will achieve minimum factors of safety provided proper construction techniques are utilized and the erosion control/revegetation blanket is properly placed and adequately attached to underlying soils. Scarification and placement of the blanket should be coordinated such that vegetation can begin to develop prior to spring runoff.

Based on our slope stability analysis and observations made during our reconnaissance visit, it is our opinion that the reclaimed fill slopes, if properly constructed, will perform similar to or better than the natural existing slopes in the area of the proposed Tank Seam Portal Access Road.

Again, we are pleased to be able to assist you with the expansion of the Bear Creek Mine. If you have any questions concerning this report, our previous reports, or if we can assist you in any other way please call at your earliest convenience.

oOo

The following Plates are attached and complete this report:

Plate 1 - Plot Plan
Plate 2 - Reclaimed Slope Cross-Section
Plate 3 - Reclaimed Slope Scarified Section

LICENSED PROFESSIONAL ENGINEER
93-180048-2202
RUSSELL L.
OWENS
STATE OF UTAH

Sincerely,

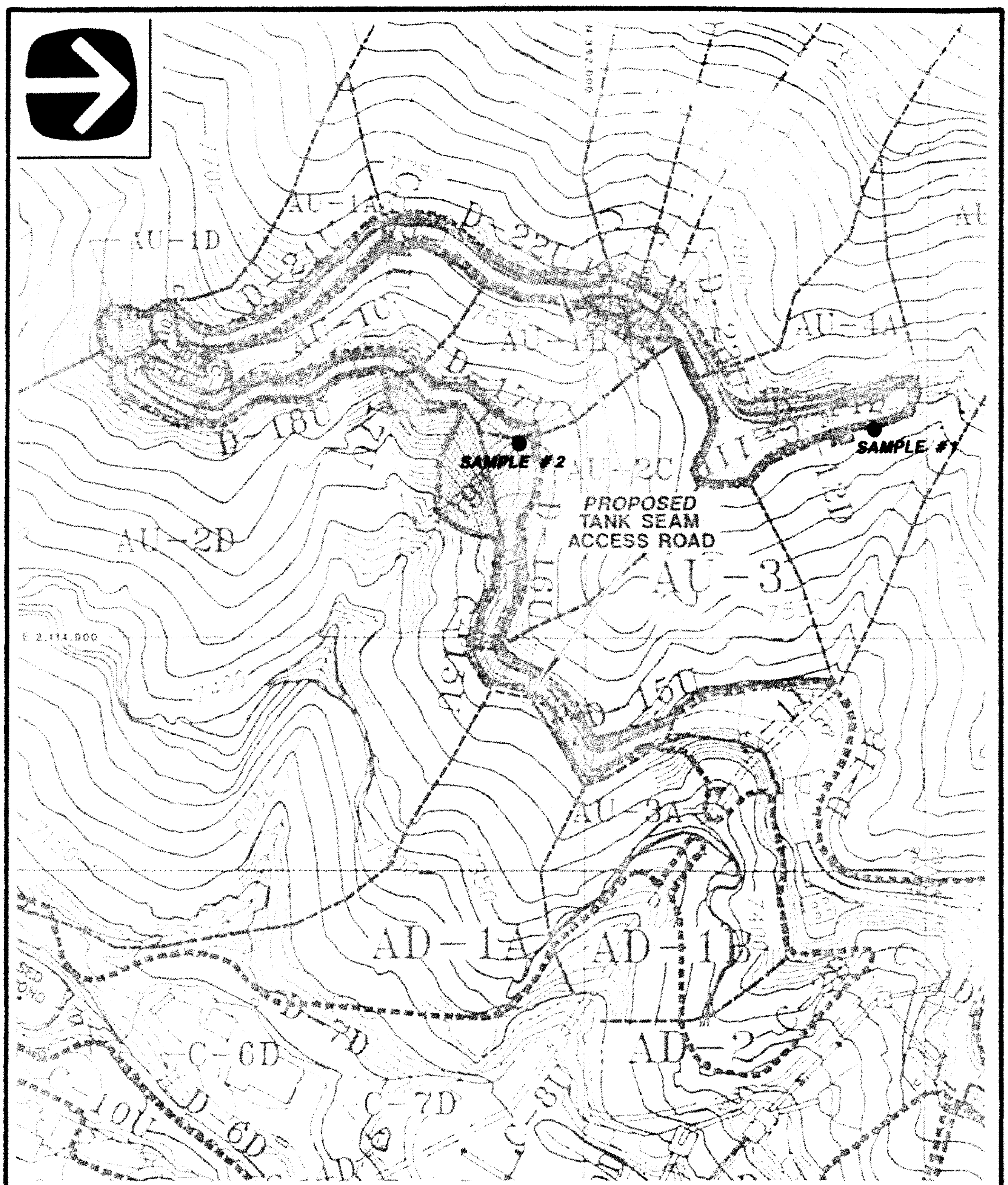
DAMES & MOORE, INC.

C. Charles Payton

C. Charles Payton, C.E.G.
Associate

Russell L Owens

Russell L. Owens, P.E.
Professional Engineer
State of Utah



100 0 100 200
SCALE IN FEET

AD-1 INDICATES DRAINAGE AREAS THAT ARE DIVERTED THROUGH SEDIMENT PONDS OR ALTERNATE SEDIMENT CONTROL STRUCTURES

AU-1 INDICATES DRAINAGE AREAS THAT ARE DIVERTED BY DITCH AND/OR CULVERT TO OFF-SITE WITH LIMITED OR NO SEDIMENT CONTROL

CO-OP MINING CO.

PLOT PLAN

BEAR CANYON MINE

REFERENCE-
ADAPTED FROM DRAWING ENTITLED, "WATERSHED MAP; BEAR CANYON," BY CO-OP MINING CO., HUNTINGTON, UTAH, PLATE 7-5, DATED 5-6-91.

Dames & Moore

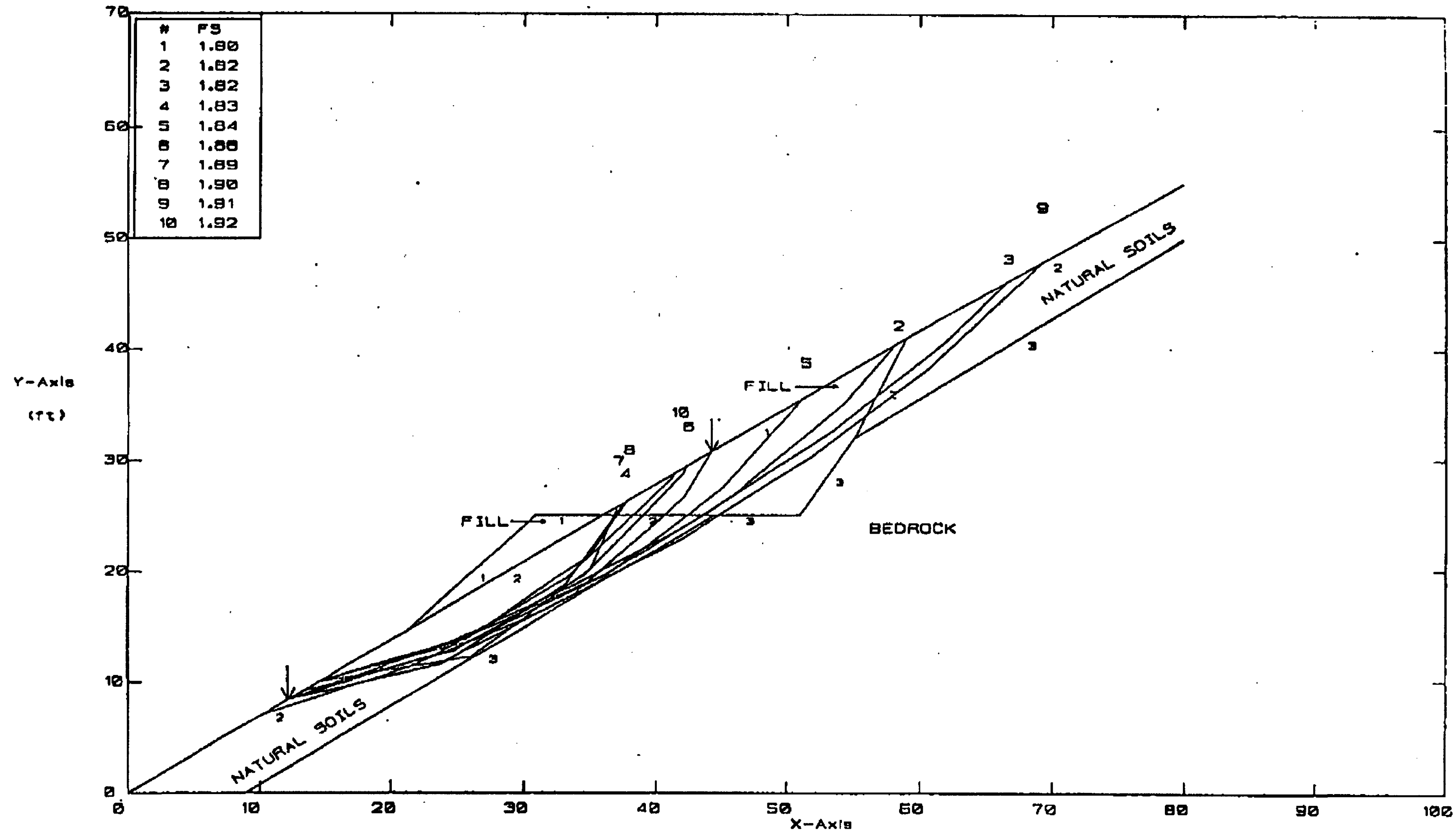
PLATE 1

FILE 27437-002-182 BY ______ DATE ______ CHECKED BY ______ DATE ______

FILE ________ BY ________ DATE ________ CHECKED BY ________ DATE ________

CO-OP Mine 1H:2V Cut slope, Reclaimed

Ten Most Critical Failure Surfaces



Soil Type No.	Total Unit Wt. (pcf)	Saturated Unit Wt. (pcf)	Cohesion Intercept (psf)	Friction Angle (deg)
1	125	130	180	36
2	120	125	180	32
3	130	140	500	38

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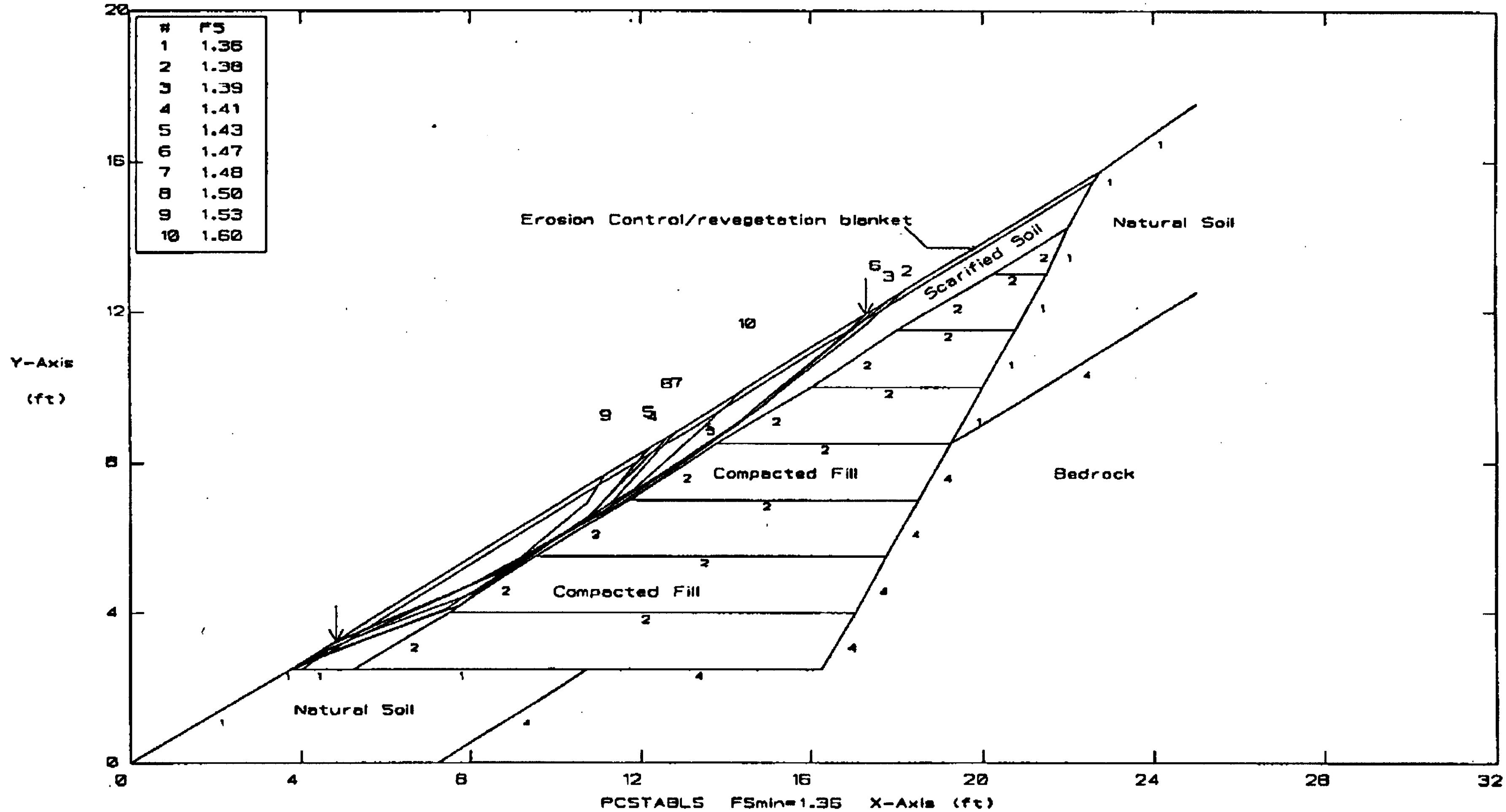
PLATE 2

FILE ______ BY ______ DATE ______ CHECKED BY ______ DATE ______

CO-OP Mine 1H:2V Cut slope, Scarified Reclaimed Slope
Ten Most Critical Failure Surfaces



Soil Type No.	Total Unit Wt. (pcf)	Saturated Unit Wt. (pcf)	Cohesion Intercept (psf)	Friction Angle (deg)	Material Type
1	120	125	180	32	Natural Soil
2	125	130	180	36	Compacted Fill
3	110	115	0	30	Scarified Soil
4	130	140	500	38	Bedrock
5	1	1	100	0	Erosion Control/Revegetation Blanket

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PLATE 3

Appendix 5-I

Cut and Fill Calculations

TABLE OF CONTENTS

Tables

CUT AND FILL CALCULATIONS

INTRODUCTION

This Appendix discusses the mass balance calculations for Reclamation Areas TS-3 through TS-9, and includes the cross-sectional representations in order to demonstrate that there is adequate volumes of fill and soil material for projected reclamation and revegetation plans. The reclamation areas are depicted on Plates 2-3. A full discussion of substitute topsoil (plant growth material) material is found in R645-301-224. Revegetation is discussed in Chapter 3.

Plates 5-6 show the contours which these cross-sections are based on. Much of the final contours on Plate 5-6 are at five and ten ft. intervals. In flatter areas contours have been drawn at 2 ft. intervals. Although every effort has been made to accurately depict final contours, the final configuration may vary according to the equipment used and the abilities of the operator. In all cases the final contours will meet safety and reclamation standards, and provide for adequate and stable drainage.

In some parts of TS-7 slopes greater then the maximum stable slope would be required in order to fill the entire cut made for the road. In these areas the maximum stable slope of 1.5H:1V will be used starting at the outer base of the berm and extending as far up the initial road cut as it can. In some of the cross-sections a small portion along the top of the initial road cut will remain (0-8 ft) as shown on the TS-7 cross-sections 4+00 and 6+00. Cuts, which will remain, are shown on plate 5-6. All highwalls, which are located in TS-6 section 24+00, and TS-7 sections 0+00, 3+00, and 5+00, will be eliminated during reclamation.

COAL WASTE VOLUME (TS-5)

Coal waste is generated as a result of coal storage and sediment pond clean-out activities. This material is not suitable for substitute topsoil material, and will be covered with a minimum of 4' of other material, including 12" of substitute topsoil material. All of this material will be re-graded and covered within TS-5.

In addition, the volume of concrete to be disposed of within TS-5 has also been shown in the volumetric calculations. Concrete will be covered with a minimum of 2' of soil, including 12" of substitute topsoil material. These values have been determined from volumes shown in Chapter 2.

Calculation of Coal Waste Volume

Coal Storage Waste:

Assume one foot deep across the coal storage pad and 6" deep over the tipple yard.

Coal Storage Waste Area:

Volume at main storage	=	(1')(720')(180'+120')/2	=	108,000 cu. ft.
Volume covering tipple	=	(0.5')(45,060 sq. ft.)	=	22,530 cu. ft.
Total Volume	=	130,530 cu. ft.	=	**4,834 cu. yd.**

Sediment:

Sediment values were taken from the sediment pond designs shown in R645-301-732.210. Sediment values used for design of sediment ponds are conservative. Therefore, for the purpose of calculating waste volumes, eighty percent of these values were used. The life of the mine will be approximately twenty-seven years.

Sediment Area:

Volume (Pond A)	=	(.80)(27)(3,848 cu. ft.)	=	83,117 cu. ft.
Volume (Pond B)	=	(.80)(27)(213 cu. ft.)	=	4,601 cu. ft.
Volume (Pond C	=	(.80)(27)(126 cu. ft.)	=	2,722 cu. ft.
Total Volume	=	90,440 cu. ft	=	**3,350 cu. yd.**

Concrete:

Volume = 8,041 cu. yd. (from Section 3.6.8)

Total Wash Volume:

Volume of Coal Waste	=	4,834 cu. yd.
Volume of Sediment	=	3,350 cu. yd.
Volume of Concrete	=	8,041 cu. yd.
Total Volume	=	**16,503 cu. yd**

SUBSTITUTE TOPSOIL MATERIAL

The substitute topsoil material that will be used is included in the cut volumes. Chapter 2, R645-301-224, describes the source of all substitute topsoil and it's use during reclamation.

CUT AND FILL VOLUMES

Areas TS-3, TS-4, and TS-9 will balance within themselves and no material will be hauled in or out of the Areas. Fill material generated in Areas TS-5 and TS-6 will be hauled to Areas TS-7 and TS-8 and the four areas will balance together. A total additional volume of 14,948 cu. yd. will be needed in TS-7 and TS-8.

Cut and fill volumes were measured using "Quicksurf" Version 4.0 3-D modeling software package, copyright 1991, Schreiber Instruments, Inc. Volumes are based on the contours on Plates 5-6 and Plates 5-2. Quicksurf was also used to generate the cross-sections which were then used to calculate the amount of substitute topsoil that was generated with each of the cuts.

Table 5I-1 shows the summarized cut and fill volumes for each section. The following pages show detailed cut and fill tables as well as cross-sections for each area. Plates 5-6 and 2-3 show the location of the cross-sections for areas TS-3 through TS-9. R645-301-240 describe the topsoil depths and there sources.

Table 5I-1 - Cut and Fill Summary
Areas TS-3 Through TS-9

Area	Fill (-) Volume (cu. yd.)	Cut (+) volume (cu. yd.)	Excess Volume (cu. yd.)[1]
TS-3	1,454	1,468	14
TS-4	3,460	3,473	13
TS-5	25,157	40,585	15,428
TS-6	5,573	8,126	2,553
TS-7	18,037	6,445	-11,592
TS-8	7,022	3,666	-3,356
TS-9	5,851	5,889	38
Cumulative Balanced Volume (cu. yd.) =			3,098

[1] An excess volume of 3,098 cu. Yds. will be generated based on the contours shown on Plates 5-6. This excess is generated in Reclamation Area TS-5, and demonstrates that there is adequate material for reclamation. During reclamation, actual contours in TS-5 can be varied in the areas of cut to eliminate this excess cut. This excess material may also be used to cover any soil found to be unsuitable at the time of reclamation.

TS-3 Sediment Pond B and Scale House Pad

To reclaim Sediment Pond B, the West embankment will be removed and used as fill material. Additional substitute topsoil material will come from the removal of culvert C-10U.

To reclaim the Scale House Pad, culvert C-10U will be removed leaving the original stream and embankment, and any asphalt will be removed from the Mine Access Road. The existing Mine Access Road will be fitted with drainage controls as shown in Appendix 7-H and left in place for post-mining access. The remaining areas within the Scalehouse Pad will be ripped and the existing substitute topsoil material will be used in place. A summary of the cut and fill volumes is shown in Table 5I-2.

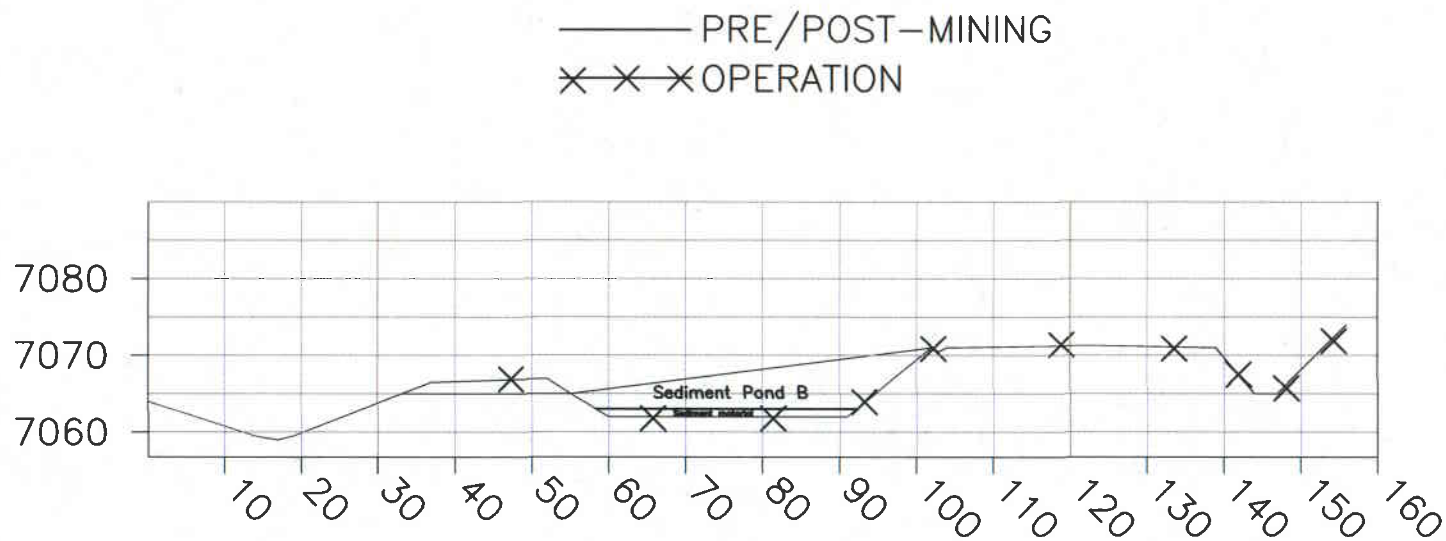
Table 5I-2 - Area TS-3 Cut & Fill Summary

	Fill (-) Volumes (cu. yd.)	Cut (+) Volumes (cu. yd.)			**Volume Cumulative (cu. yd.)**
Section	**Total Fill Volume**	Substitute Topsoil	Regular Soil	**Total Cut Volume**	
A-A	**1054***	190	0	**190**	**-864**
B-B	**400**	668	610	**1,278**	**14**
Totals	**1,454**	858	610	**1,468**	

* It is assumed that sediment Pond B would contain 82 cu. yd. of sediment at the start of reclamation. The actual volume may vary slightly.

SECTION A–A

SEDIMENTATION POND "B"

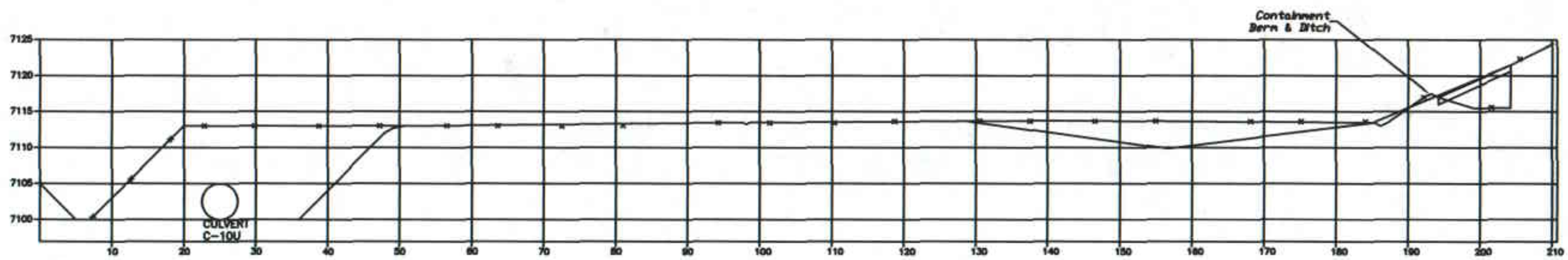


SECTION B–B

SCALE HOUSE PAD CULVERT

PRE–MINING/POST–MINING

OPERATION



TS-4 Sediment Pond A

The embankment on each side of the Sediment Pond will be cut to original contour. The fill generated will be used as fill material and substitute topsoil material. The cut and fill amounts within TS-4 will balance.

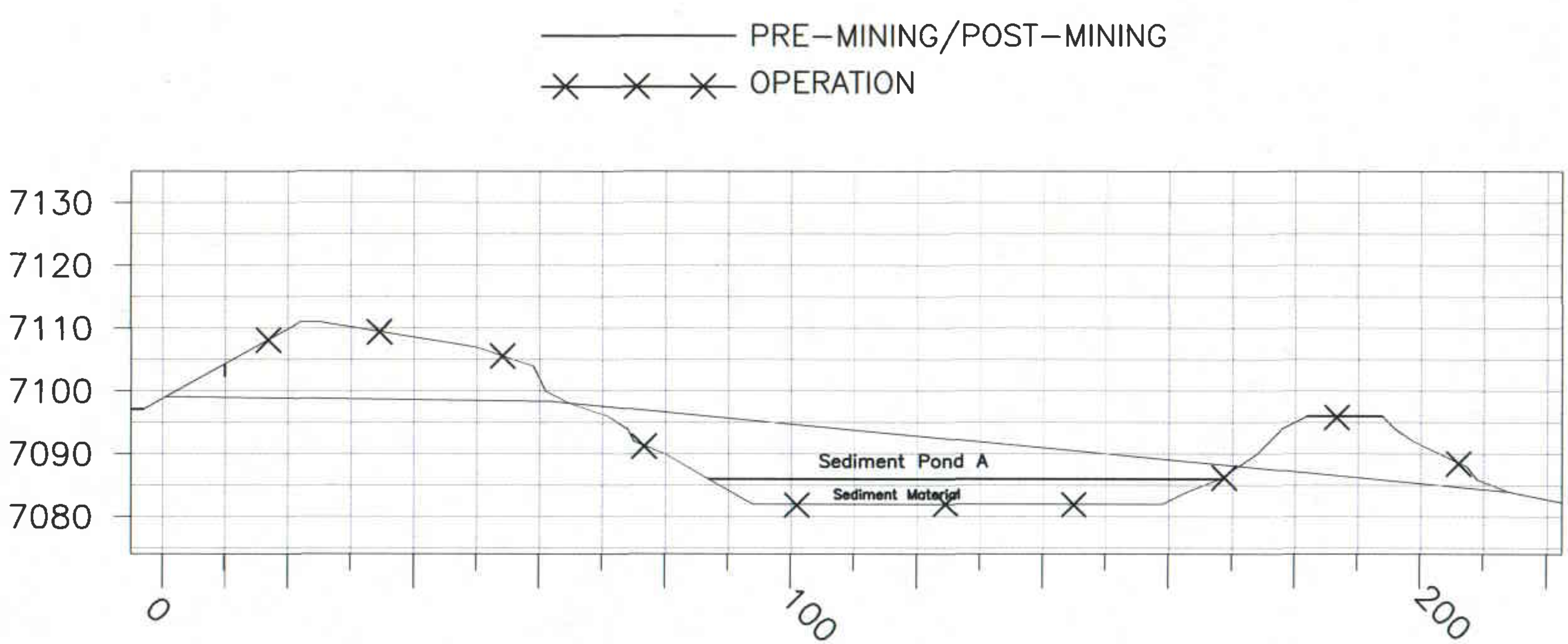
Table 5I-3 - Area TS-4 Cut & Fill Summary

	Fill (-) Volumes (cu. yd.)	Cut (+) Volumes (cu. yd.)			**Volume Cumulative (cu. yd.)**
Section	**Total Fill Volume**	Substitute Topsoil	Regular Soil	**Total Cut Volume**	
C-C	**3,460**	1,008	2,465	**3,473**	**13**

* It is assumed that Sediment Pond A would contain 878 cu. yd. of sediment at the start of reclamation. The actual volume may vary slightly.

SECTION C–C

SEDIMENT POND "A"



TS-5 Tipple and Load-out Area

The tipple and load-out area will be reclaimed to match the contours shown on Plate 5-6C, although actual contours may vary somewhat to account for required cut volumes and onsite concrete disposal. The excess coal waste will be used as fill in this area and will be buried a minimum of 4' deep, with a minimum of 12" of substitute topsoil material applied on the surface. In sections 3+00 through 8+00 the existing road on the east side will be left in place for post mining access as shown on Plate 5-6C. Where no re-grading is required, and in areas were the cut leaves at least 12" of substitute topsoil material, the area will be ripped and existing substitute topsoil material will be used in place. 15,428 cu. yd of material can be generated in TS-5 for use in TS-7 and TS-8, which exceeds the volume needed as shown in Table 5I-1. 1,000 yds^3 of this material will go to TS-17 as described on page 5K-7.

The west slope of the tipple area, shown in cross-section 9+00, will be filled to cover the coal waste which exists in the area. Soil and substitute topsoil from the coal storage pad will then be placed over the coal waste as shown in cross-section 9+00. The slope below the tipple pad will consist of removing coal waste material and replacing it with substitute topsoil material. Although the removing and replacing results in a minimal change in the cross-section and contours, the volumes in Table 5I-4 reflect the removal and replacement of this material.

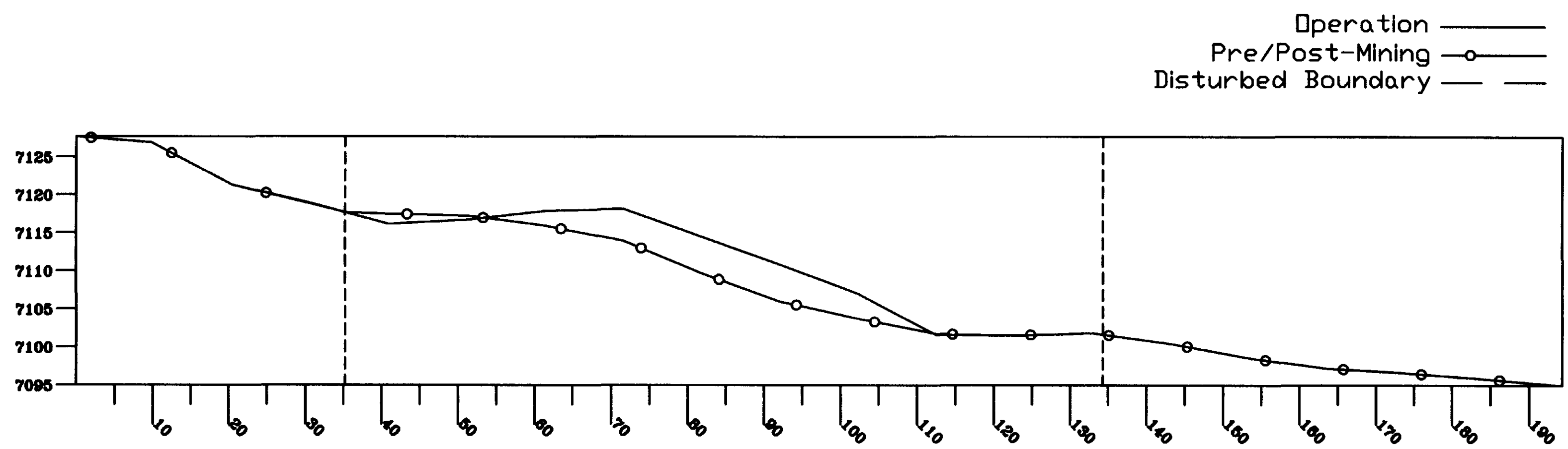
A summary of the cut and fill volumes is shown in Table 5I-4.

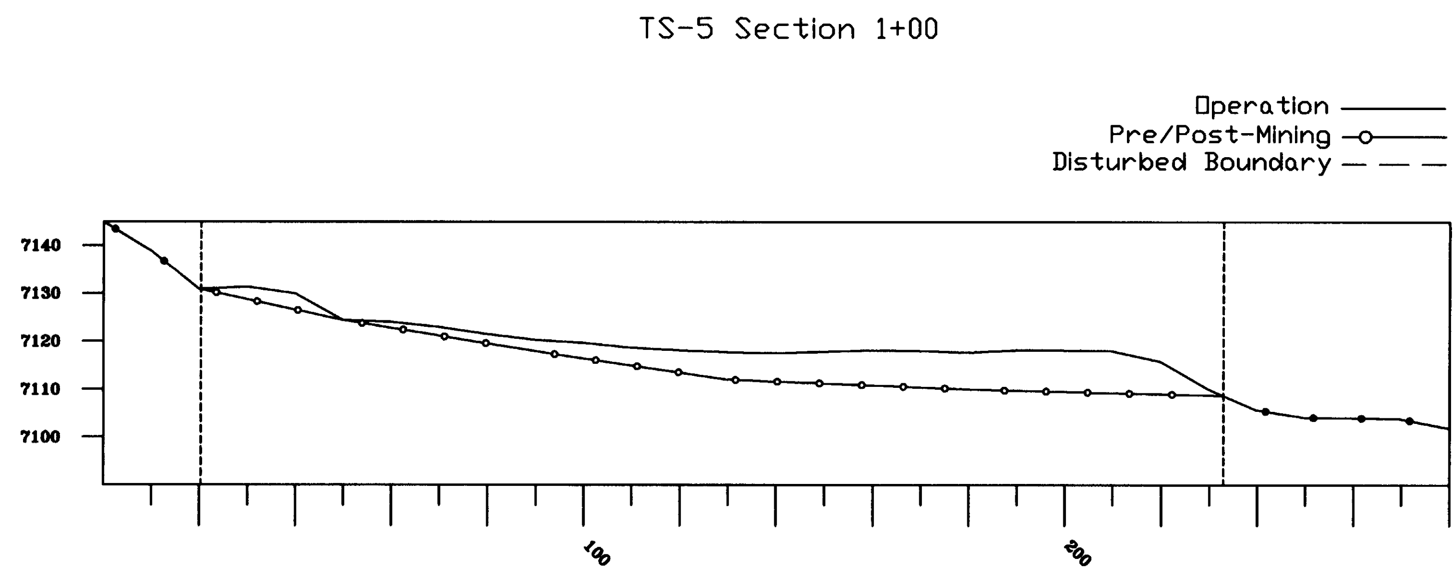
Table 5I-4 - Area TS-5 Cut & Fill Summary

	Fill (-) Volumes (cu. Yd.)					Cut (+) Volumes (cu. yd.				**Volume**
Section	Min. required[1]	Soil	Coal Waste / Sediment	Concrete	**Total Fill Volume[2]**	Substitute Topsoil	Other Soil	Coal Waste / Sediment	**Total Cut Volume**	**Cumulative (cu. yd.)**
0+00	0	89	0	0	**89**	304	407	0	**711**	**622**
1+00	0	44	0	0	**44**	1,148	3,311	0	**4,459**	**5037**
2+00	452	690	0	210	**690**	452	100	0	**552**	**4899**
3+00	385	596	209	325	**805**	359	189	0	**542**	**4642**
4+00	741	1,182	813	590	**1,995**	2,351	0	412	**2,763**	**5410**
5+00	504	2,316	411	433	**2,737**	6,244	3,457	1,095	**10,796**	**13469**
6+00	1,295	1,322	430	644	**1,752**	3,246	552	1,210	**5,008**	**16725**
7+00	740	989	137	400	**1,126**	359	0	1,004	**1,363**	**16962**
8+00	911	1,326	2,726	763	**4,052**	1,937	892	1,278	**4,107**	**17017**
9+00	3,481	5,145	2,724	3,548	**7,869**	3,193	1,319	2,105	**6,617**	**15765**
10+00	452	630	667	440	**1,297**	923	32	775	**1,730**	**16198**
11+00	415	1,041	0	274	**1,041**	303	0	305	**608**	**15765**
12+00	430	755	0	293	**755**	429	90	0	**519**	**15529**
13+00	480	838	67	121	**905**	244	560	0	**804**	**15428**
Totals		16,973	8,184	8,041	**25,157**	21,492	10909	8,184	**40,585**	

[1] These volumes represent the minimum volume of soil which is required in order to cover the coal waste and concrete material to show that the fill volumes being placed are adequate to cover the coal waste and concrete material.
[2] Fill volumes do not include concrete volumes since concrete disposal on site is provided in separate bond calculations. The volumes are provided here to account for the volumes shown on the cross-sections.

TS-5 Section 0+00




TS-5 Section 1+00
Operation
Pre/Post-Mining
Disturbed Boundary
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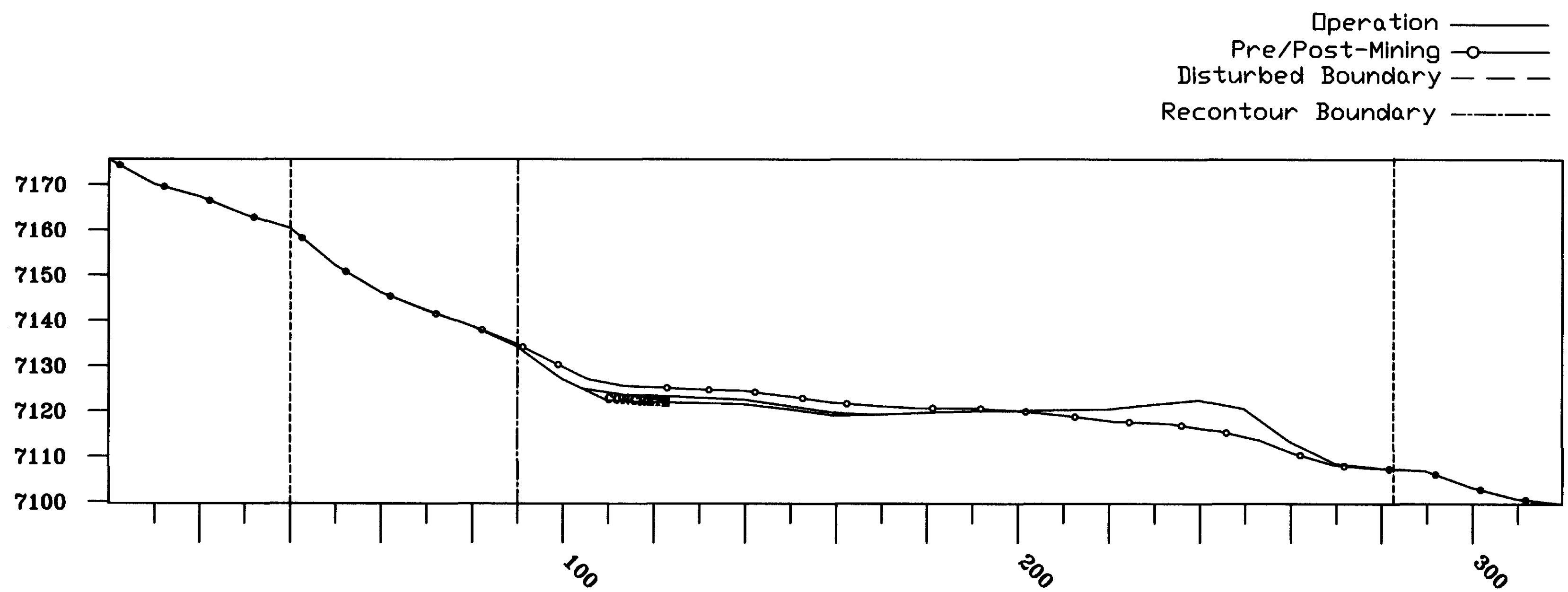
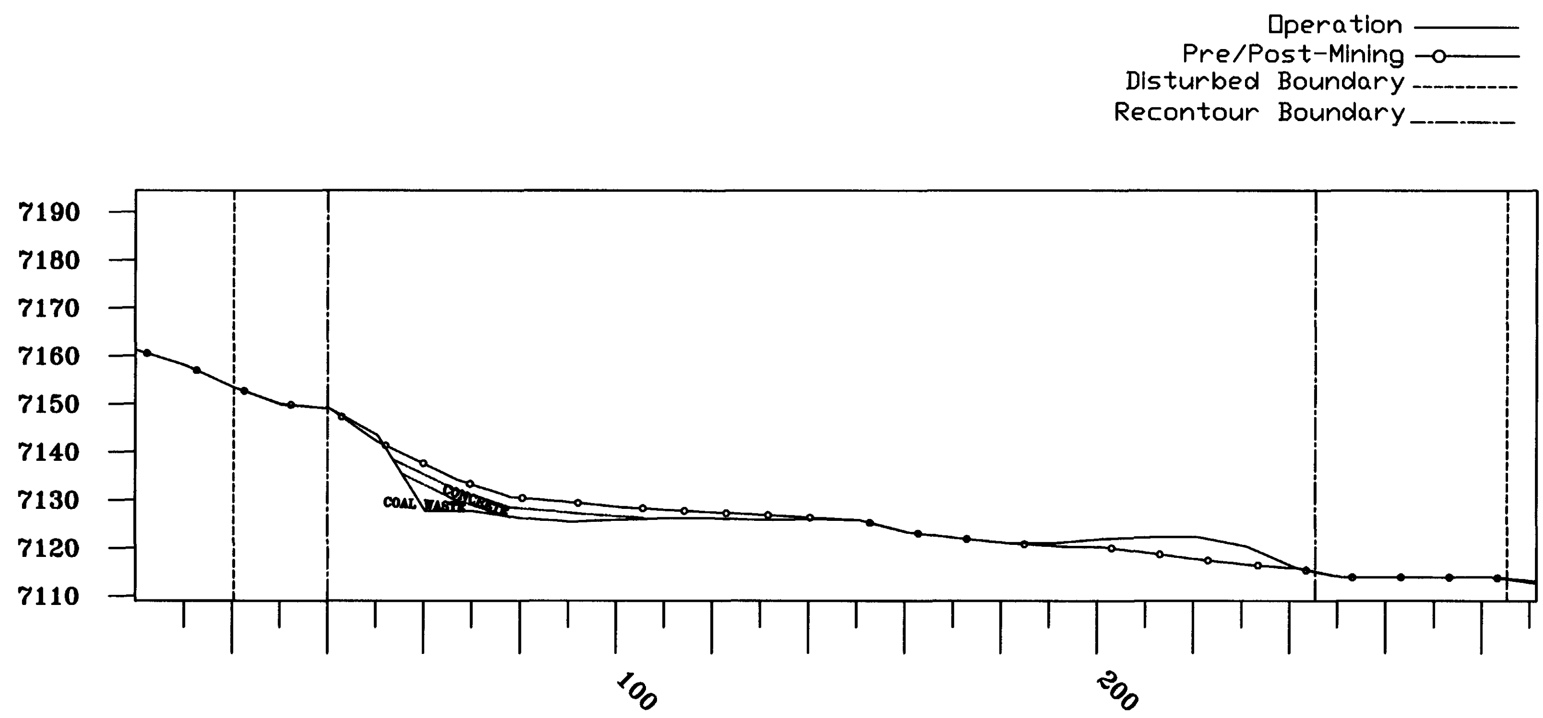
7130
7120
7110
7100
100
200

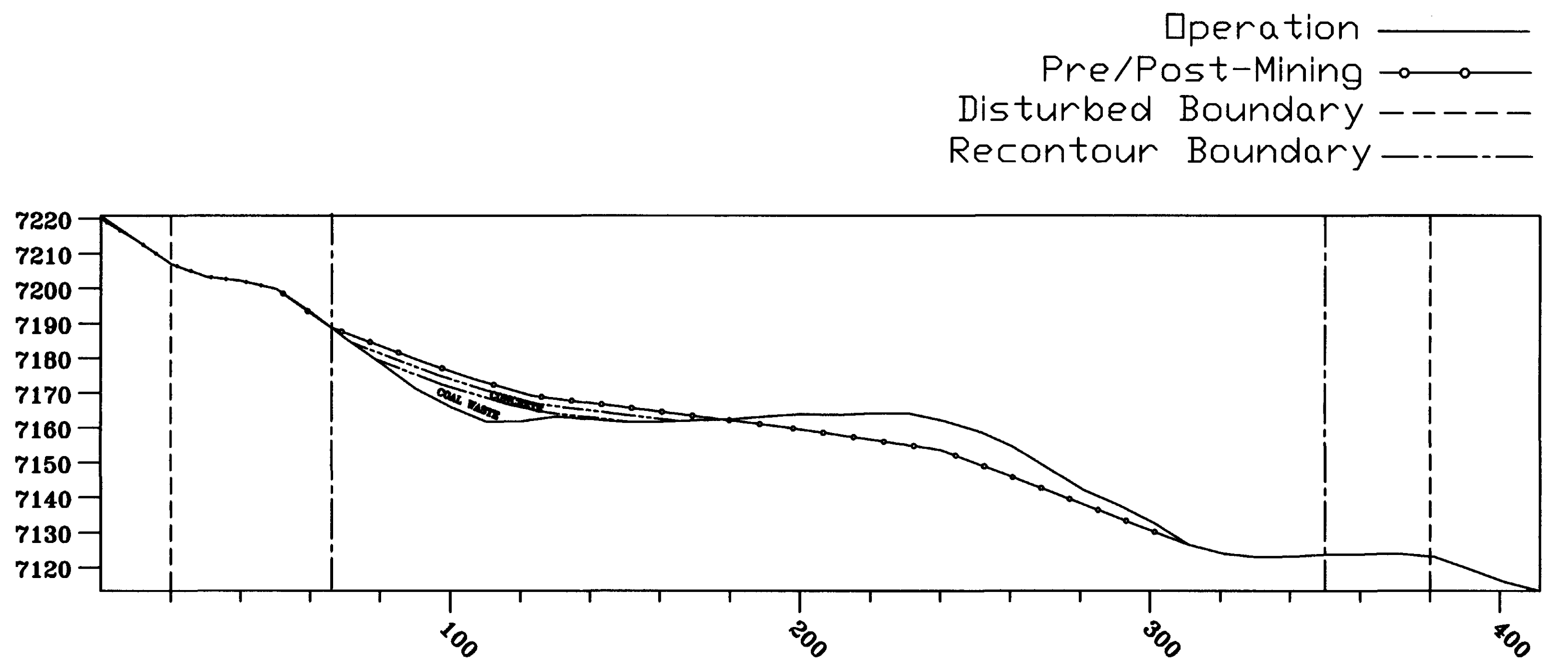
TS-5 Section 2+00



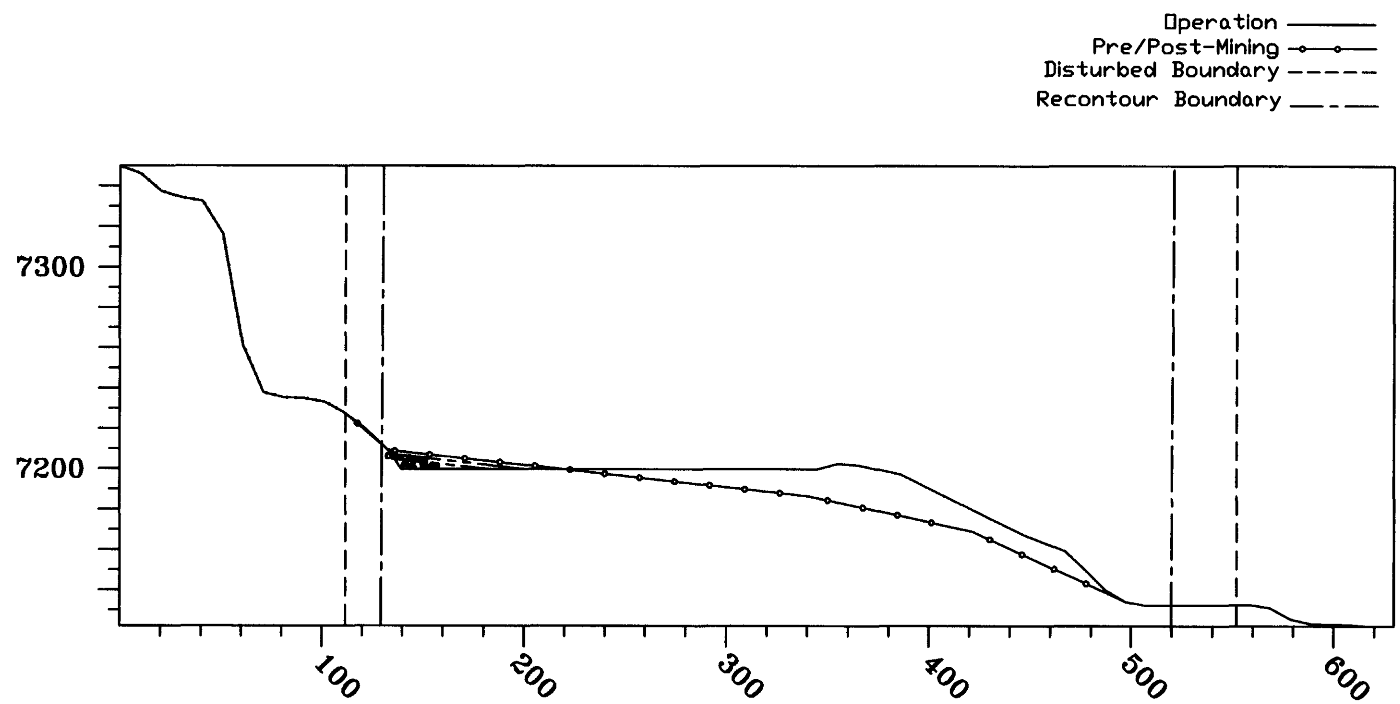
TS-5 Section 3+00



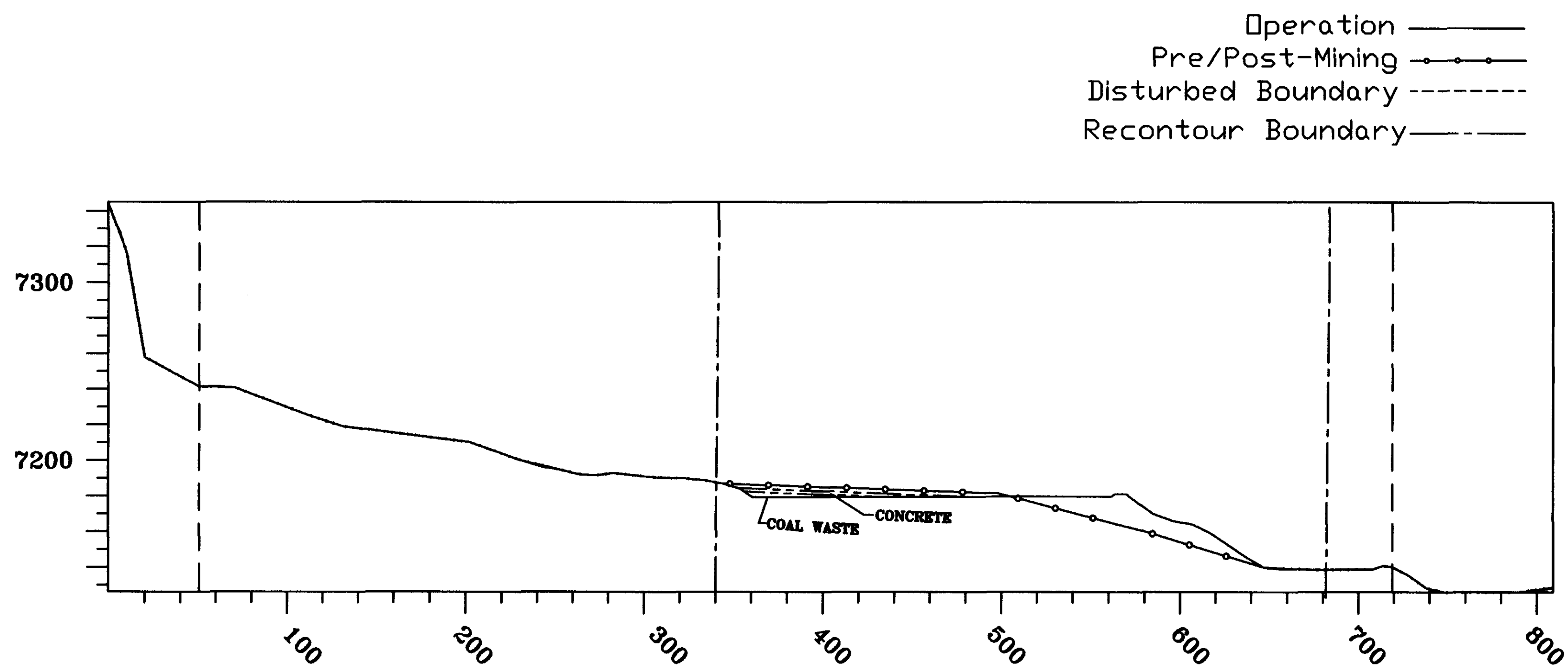
TS-5 Section 4+00



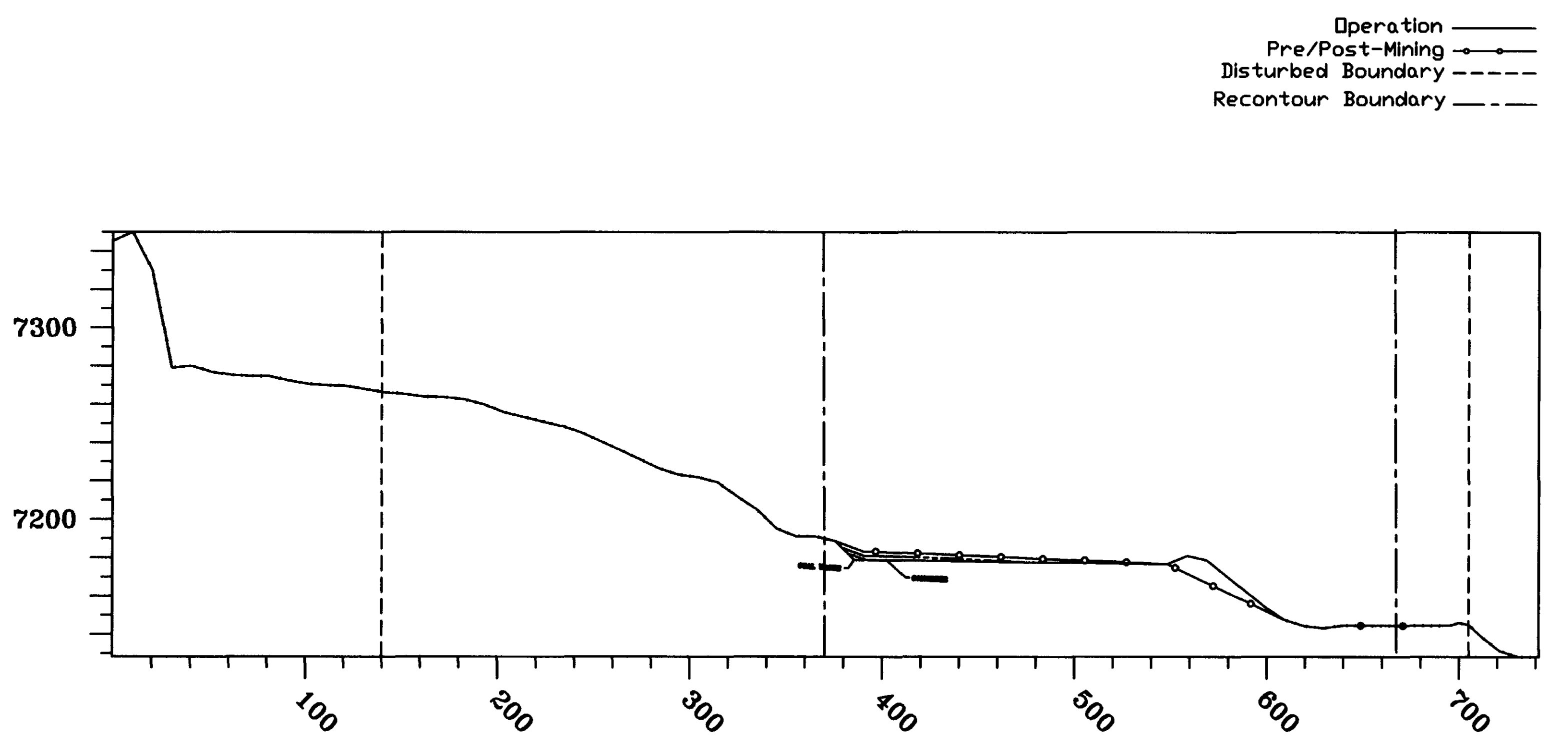
TS-5 Section 5+00



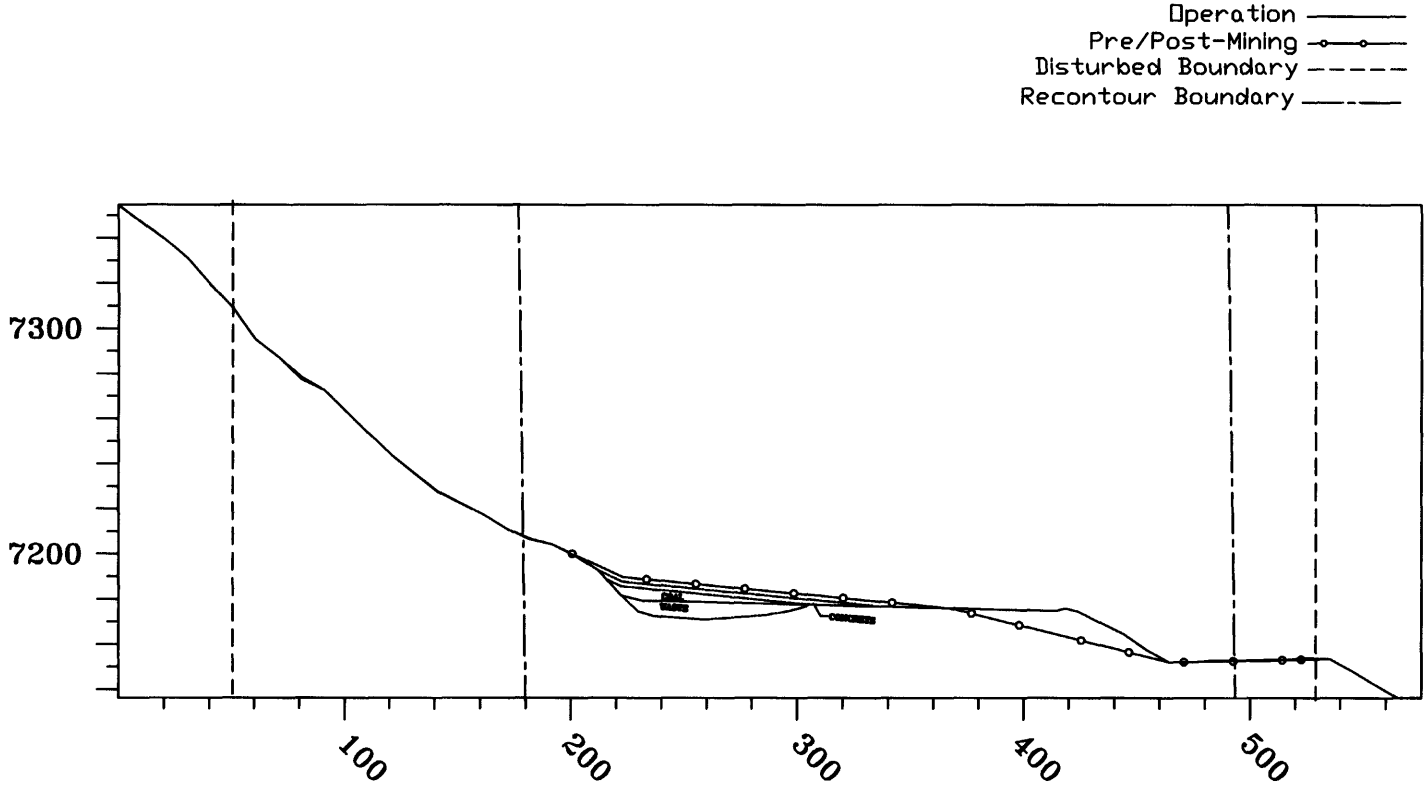
TS-5 Section 6+00



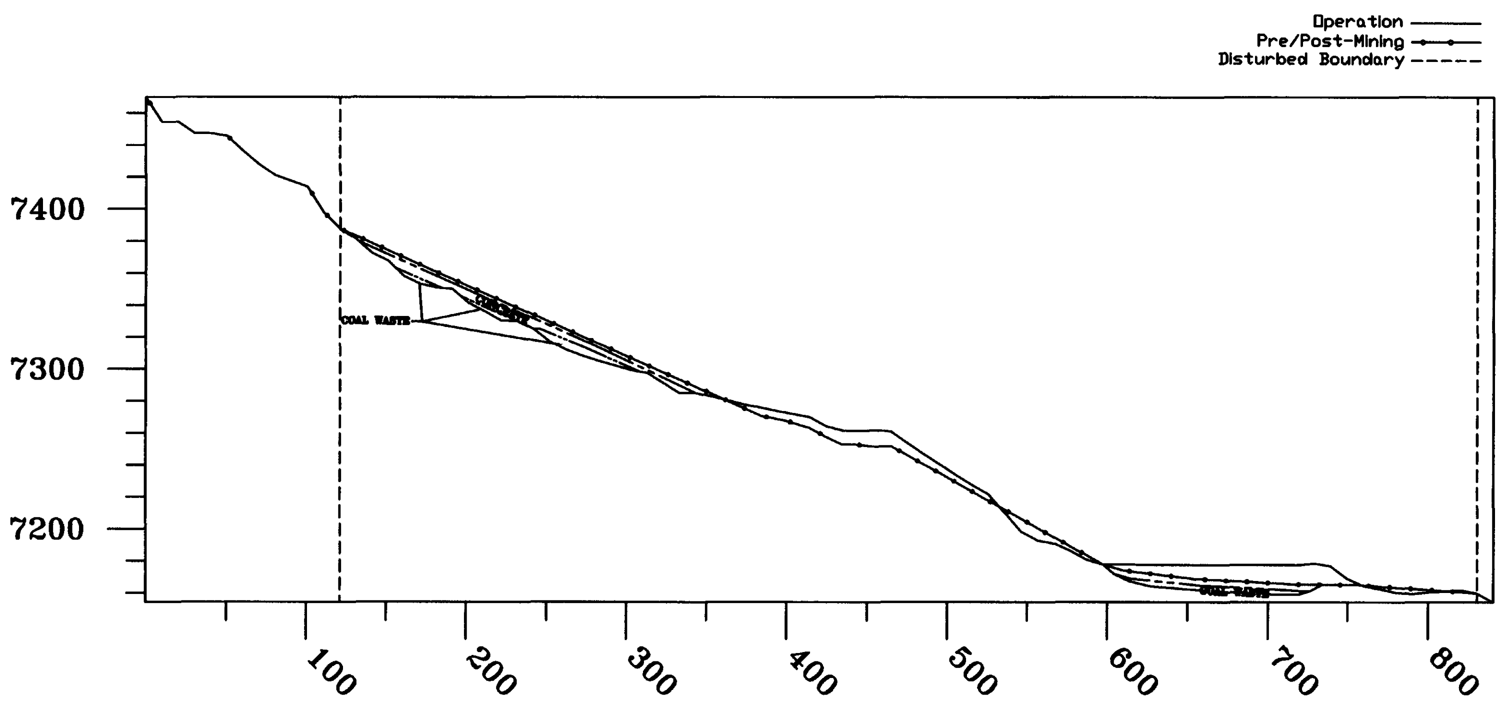
TS-5 Section 7+00



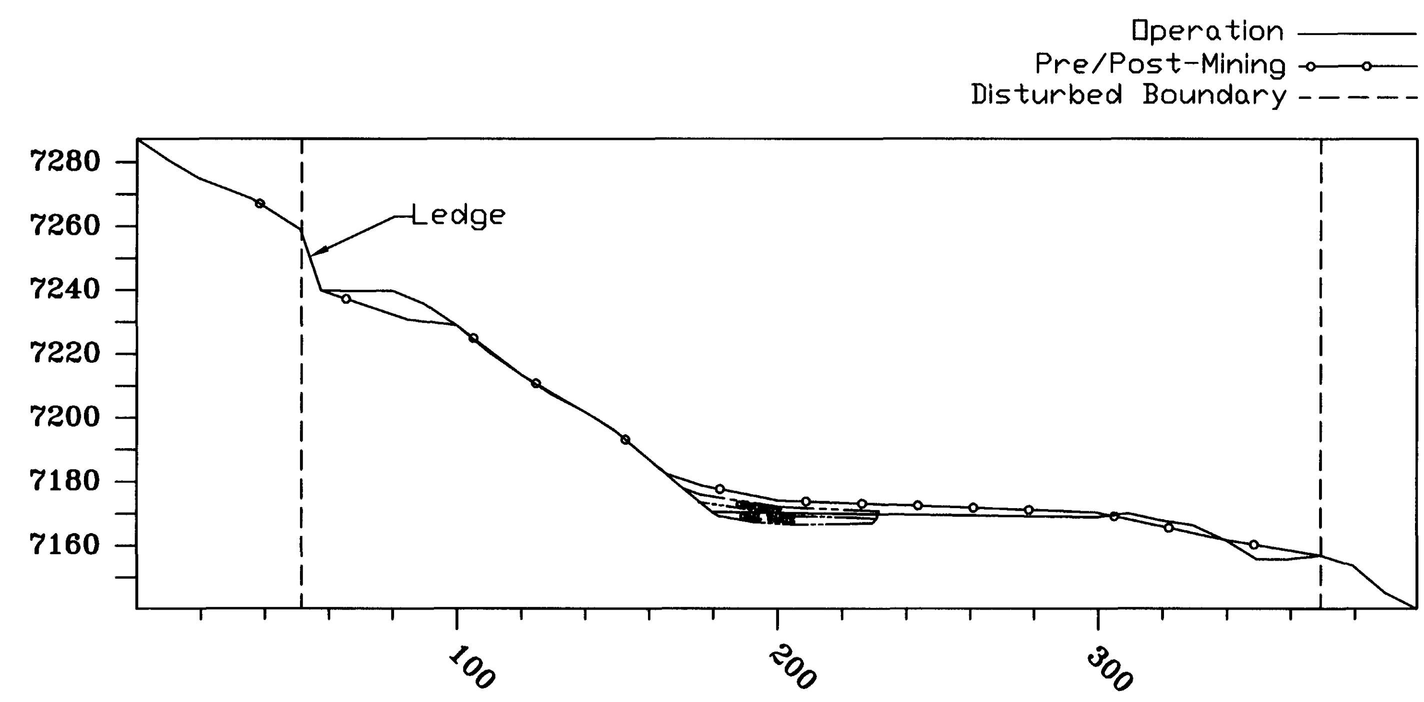
TS-5 Section 8+00



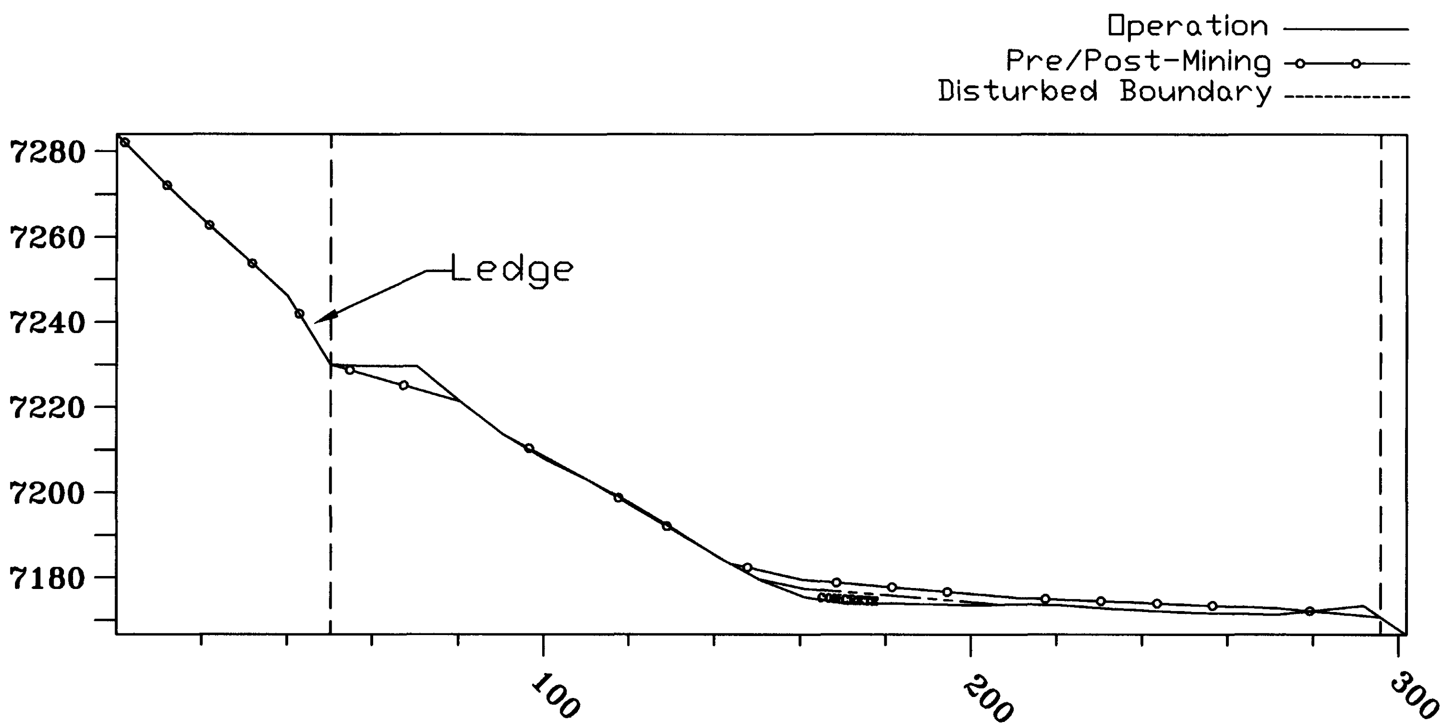
TS-5 Section 9+00



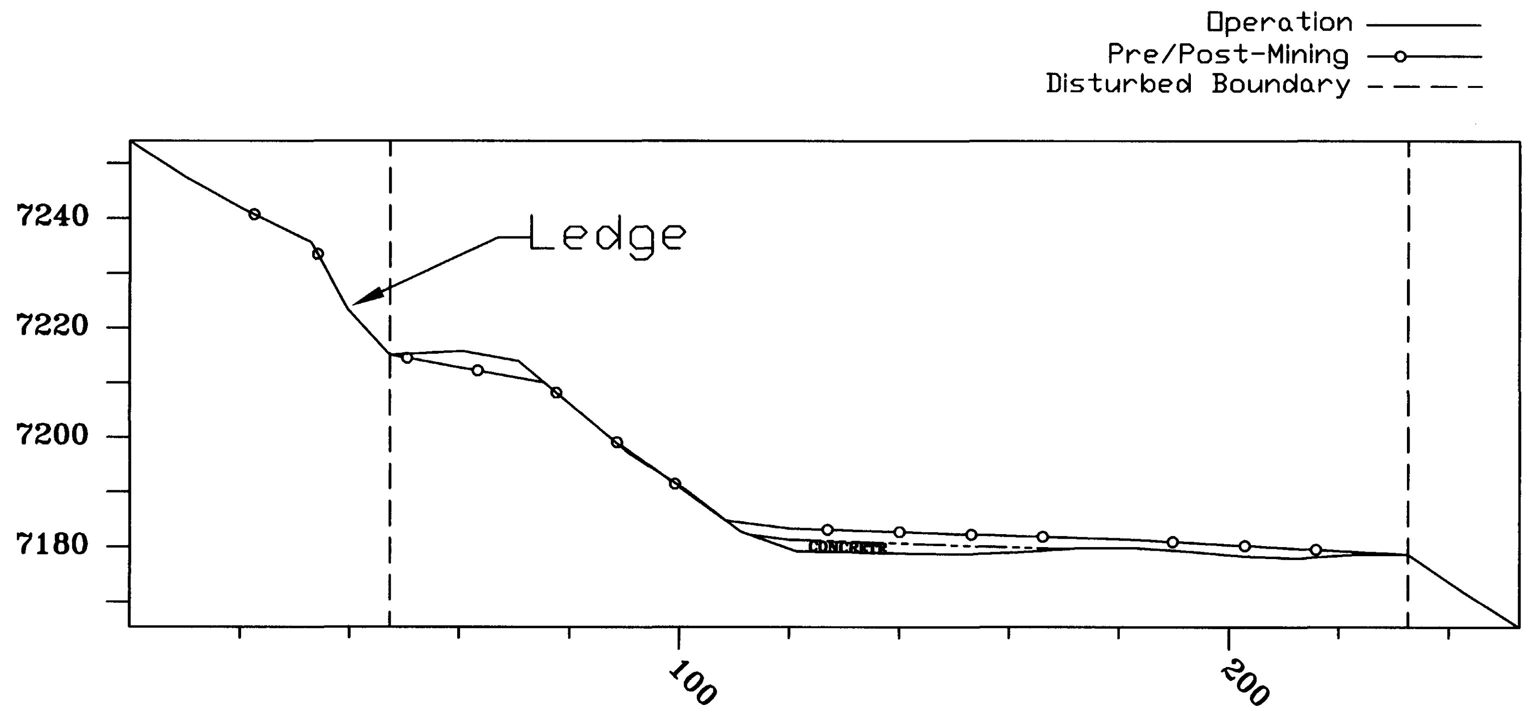
TS-5 Section 10+00



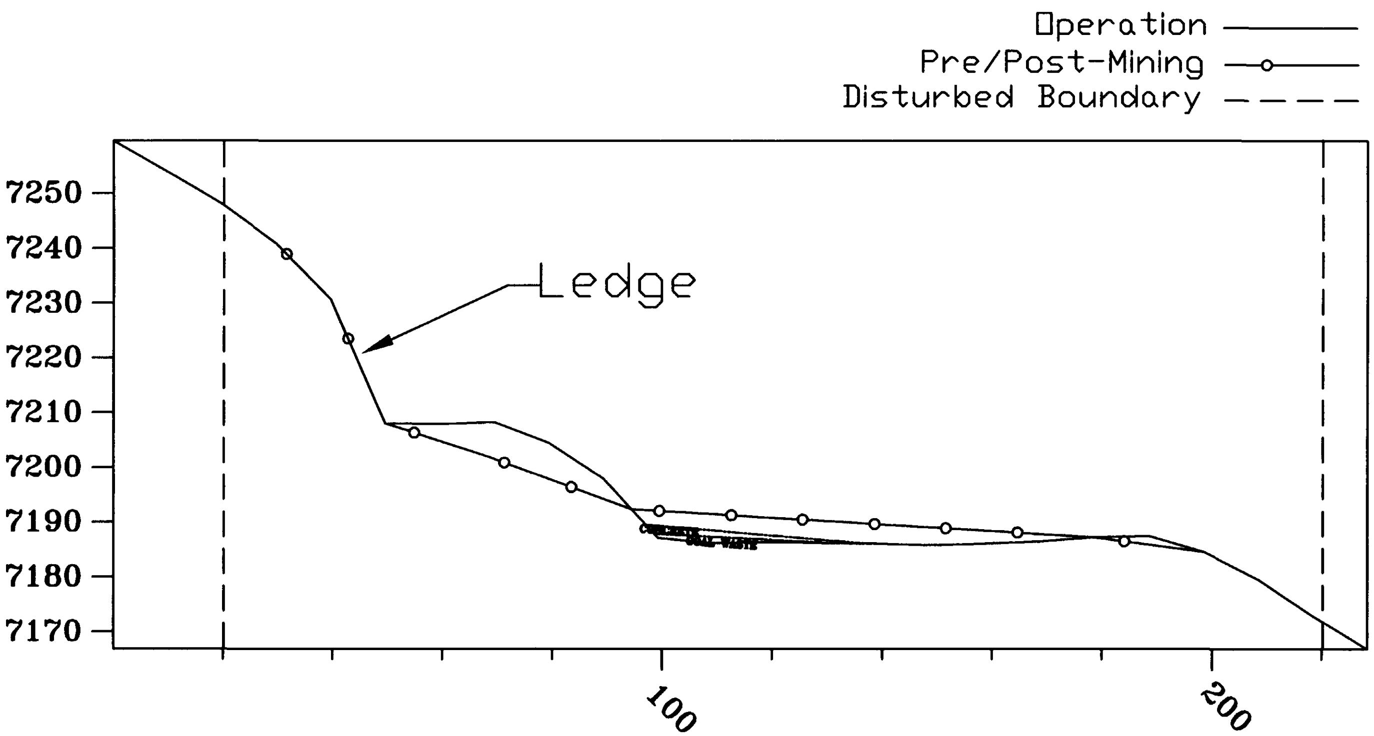
TS-5 Section 11+00



TS-5 Section 12+00



TS-5 Section 13+00



TS-6 Portal Access Road

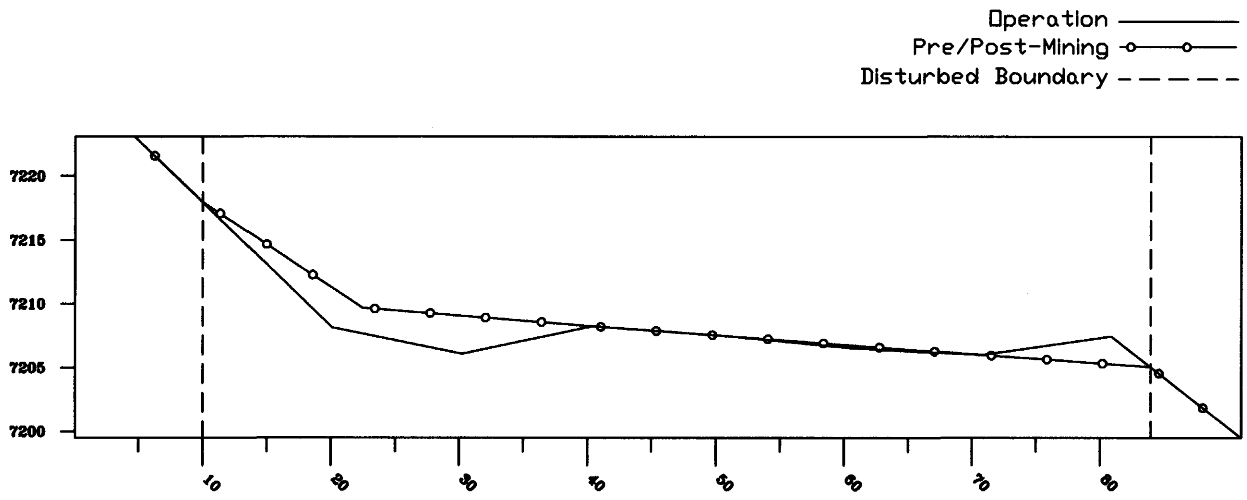
TS-6, which includes the Portal Access road, will be reclaimed as shown on the following cross-sections so that the contour matches those shown on plates 5-6C and 5-6D. Sections 11+00, 12+00, 13+00, 15+00, and 16+00 contain BTCA Area "G" which has already been reclaimed and currently supports vegetation. Sections 13+00 and 14+00 are located at the switch back. Both section 12+00 and 15+00 are south of section 13+00 but 15+00 is at a higher elevation. The reclaimed slope shown on 12+00 and 15+00 is the same one, and the reclaimed slope shown on 11+00 and 16+00 is also the same one.

2,553 cu. yd. of material will be generated in this Area for use in TS-7 and TS-8. A summary of the cut and fill values is shown in Table 5I-5.

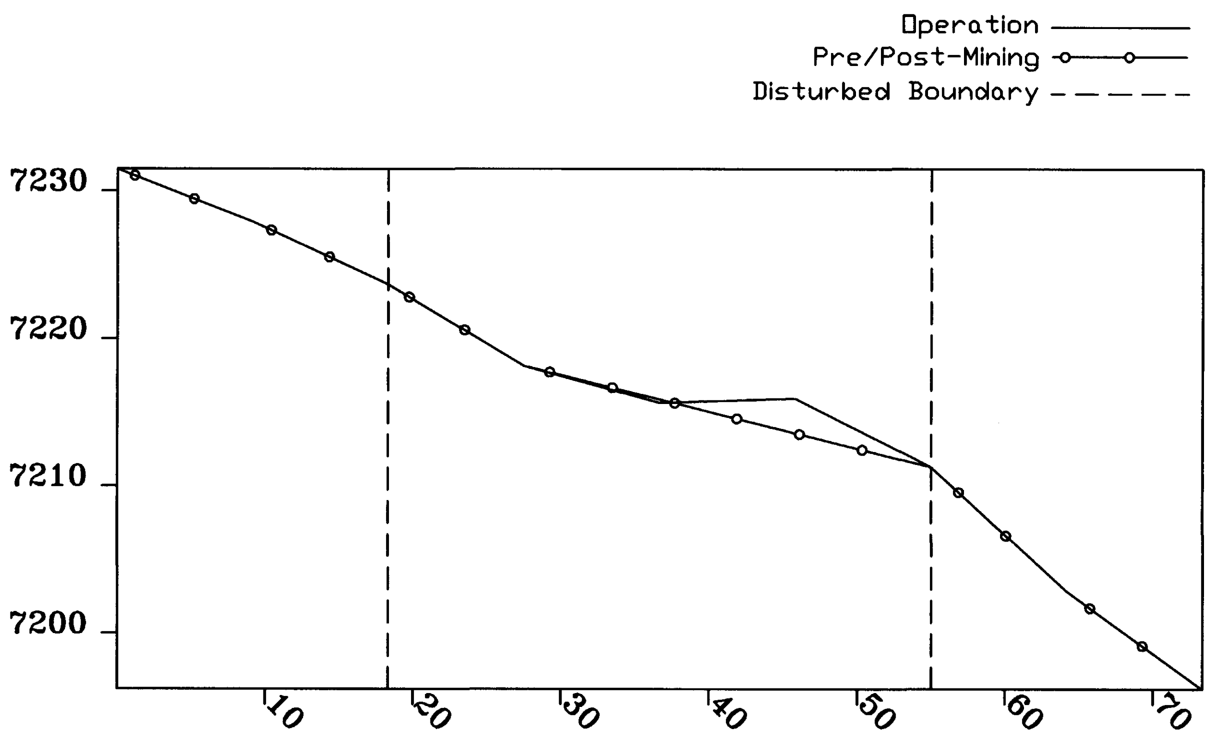
Table 5I-5 – Area TS-6 Cut and Fill Summary

	Fill (-) Volumes (cu. yd.)	Cut (+) Volumes (cu. yd.)			
Section	**Total Fill Volume**	Substitute Topsoil	Other Soil	**Total Cut Volume**	**Volume Cumulative (cu. yd.)**
0+00	**207**	52	0	**52**	**-155**
1+00	**4**	76	0	**76**	**-83**
2+00	**0**	5	0	**5**	**-78**
3+00	**82**	82	0	**82**	**-78**
4+00	**0**	274	0	**274**	**196**
5+00	**74**	74	0	**74**	**196**
6+00	**30**	30	0	**30**	**196**
7+00	**141**	144	0	**144**	**199**
8+00	**130**	52	0	**52**	**121**
9+00	**111**	63	0	**63**	**73**
10+00	**51**	167	0	**167**	**189**
11+00	**107**	107	0	**107**	**189**
12+00	**111**	111	0	**111**	**189**
13+00	**504**	104	0	**104**	**-211**
14+00	**419**	1,096	0	**1,096**	**466**
15+00	**711**	0	0	**0**	**-245**
16+00	**163**	48	0	**48**	**-360**
17+00	**0**	193	0	**193**	**-167**
18+00	**78**	226	0	**226**	**-19**
19+00	**918**	148	0	**148**	**-789**
20+00	**0**	674	268	**942**	**153**
21+00	**0**	274	0	**274**	**427**
22+00	**218**	458	0	**458**	**667**
23+00	**218**	211	0	**211**	**660**
23+50	**41**	343	33	**376**	**995**
24+00	**1,089**	37	0	**37**	**-57**
25+00	**159**	96		**96**	**-120**
26+00	**7**	1,218	538	**1,756**	**1,629**
27+00	**0**	537	172	**709**	**2,338**
28+00	**0**	211	4	**215**	**2,553**
Totals	**5,573**	7,111	1,015	**8,126**	

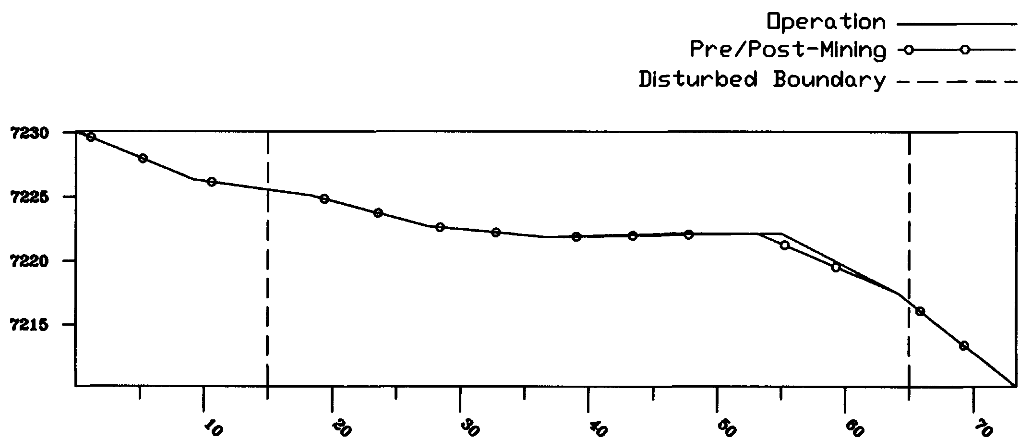
TS-6 Section 0+00



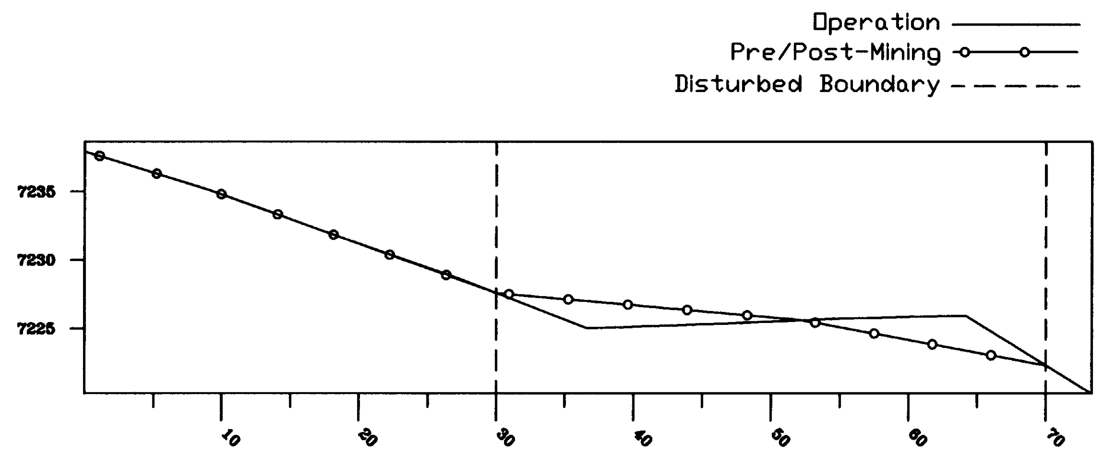
TS-6 Section 1+00



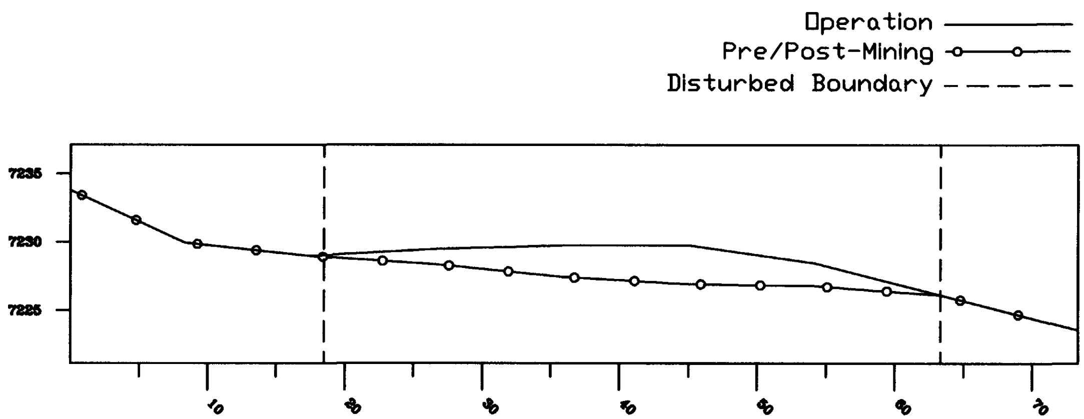
TS-6 Section 2+00



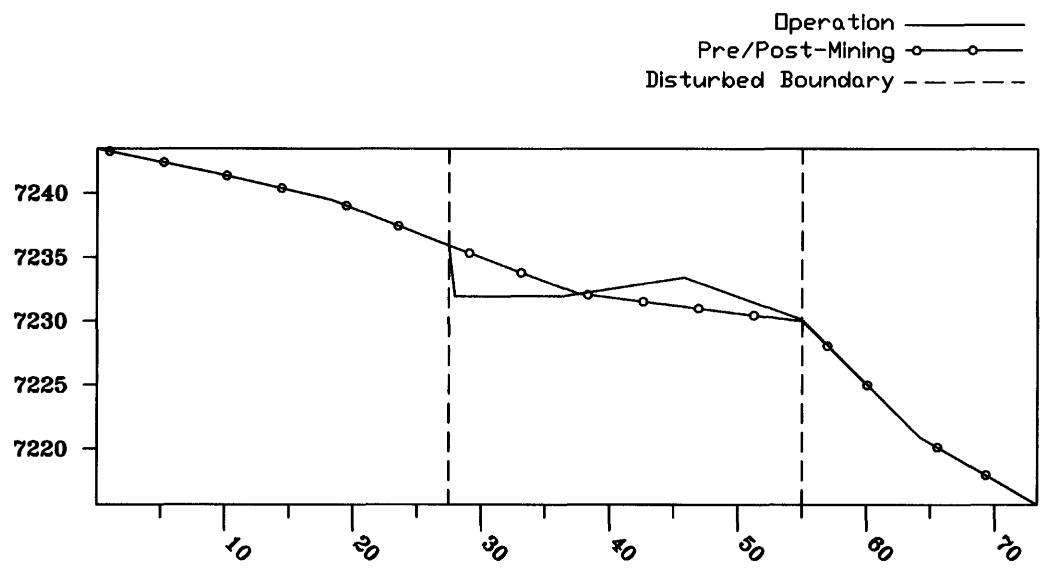
TS-6 Section 3+00



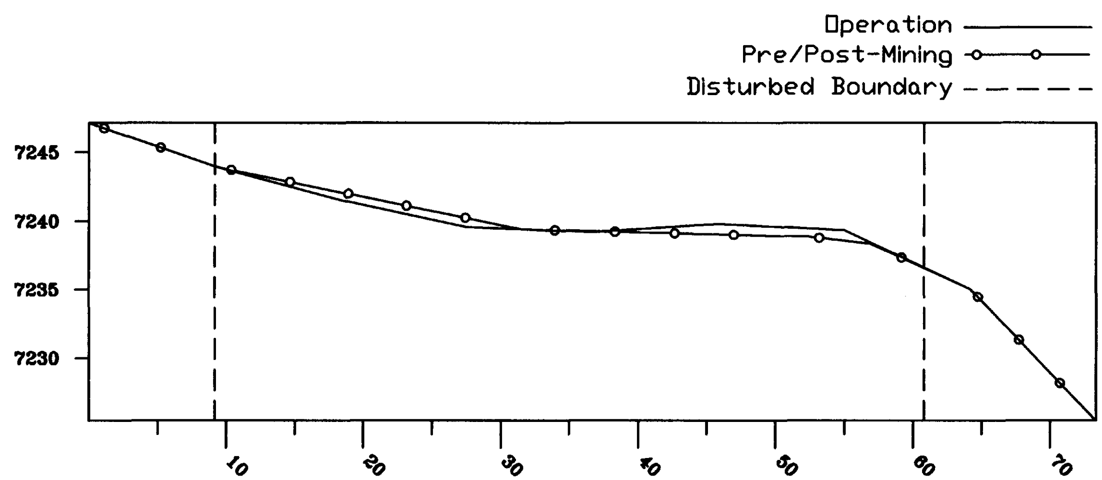
TS-6 Section 4+00



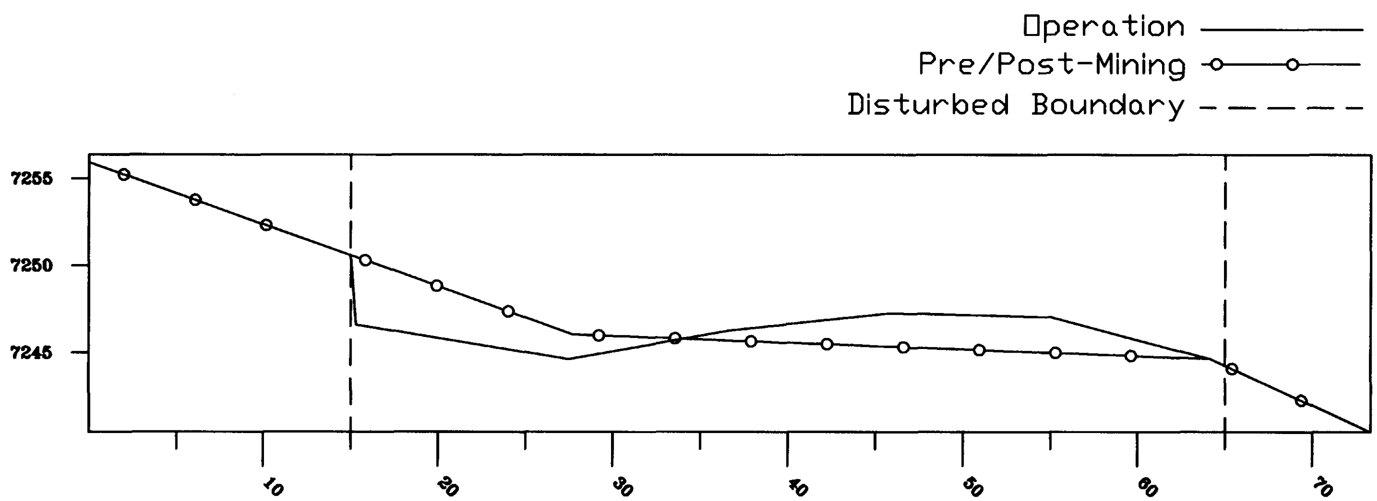
TS-6 Section 5+00



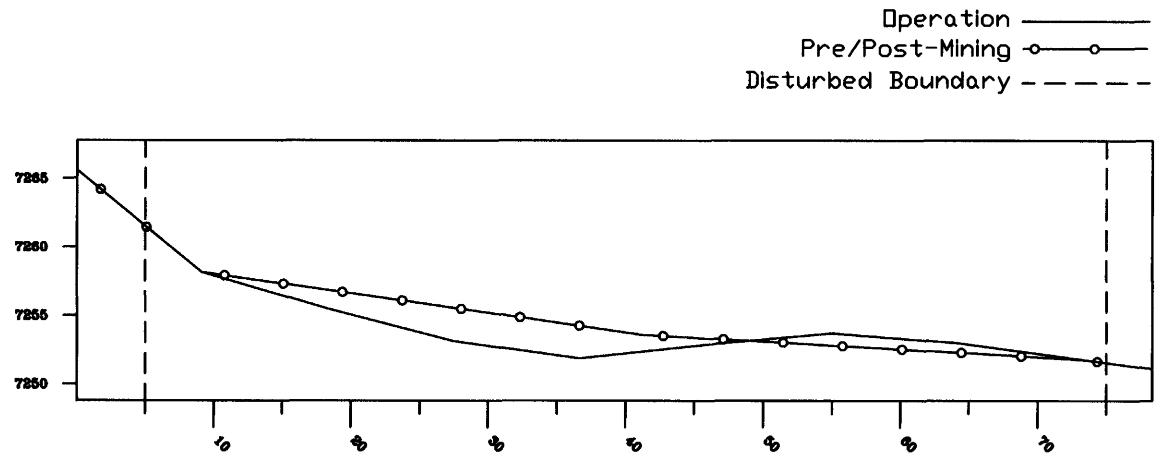
TS-6 Section 6+00



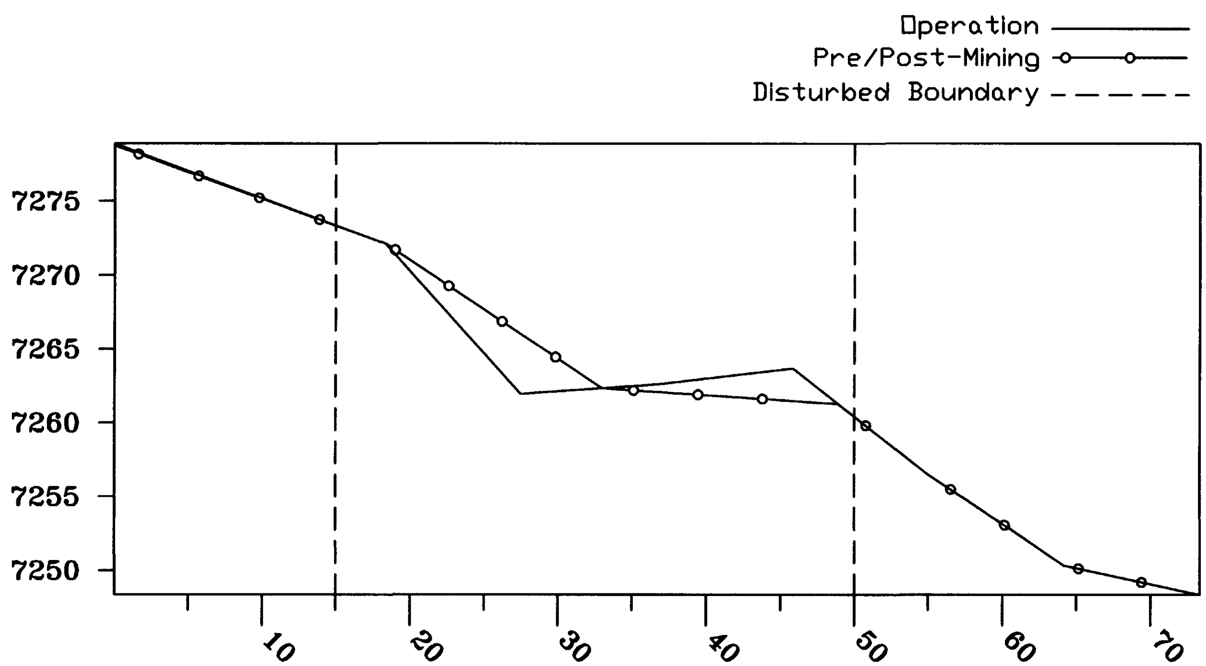
TS-6 Section 7+00



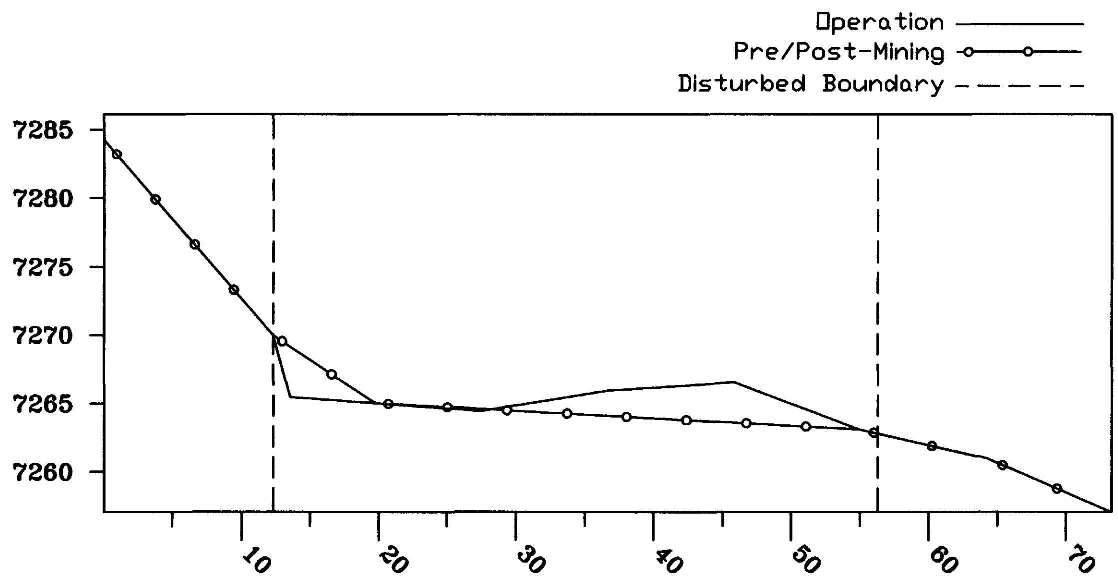
TS-6 Section 8+00



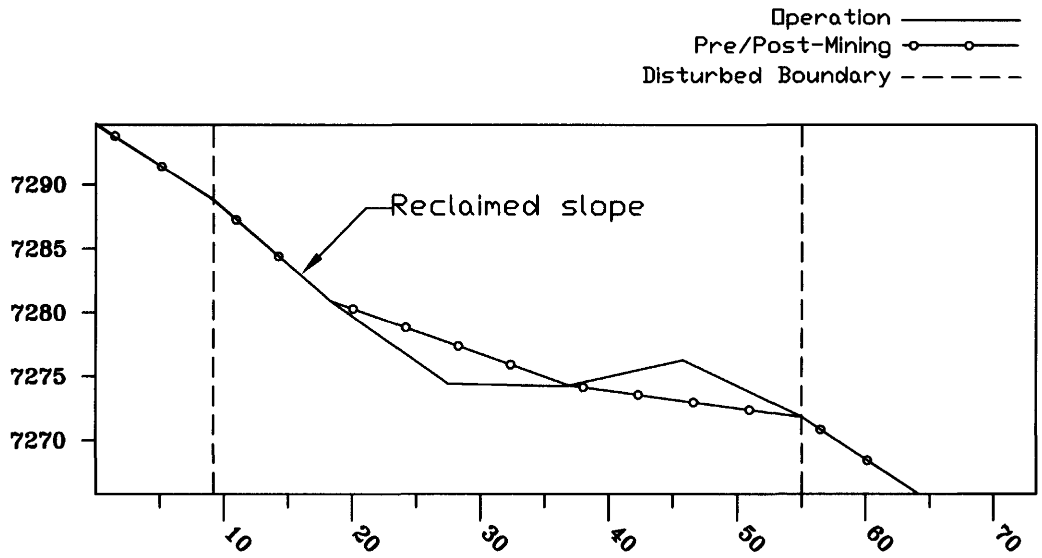
TS-6 Section 9+00



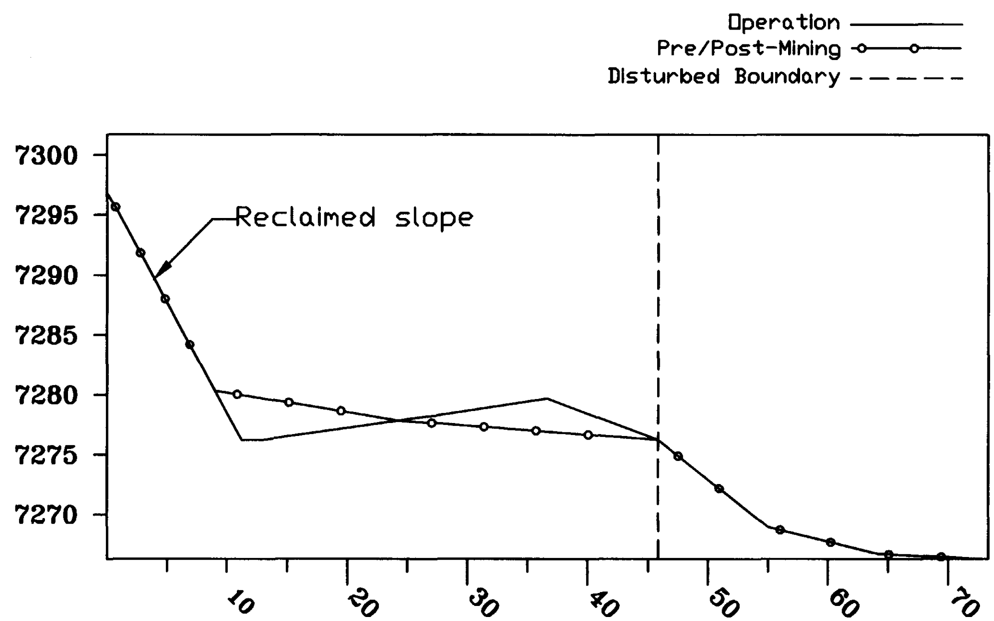
TS-6 Section 10+00



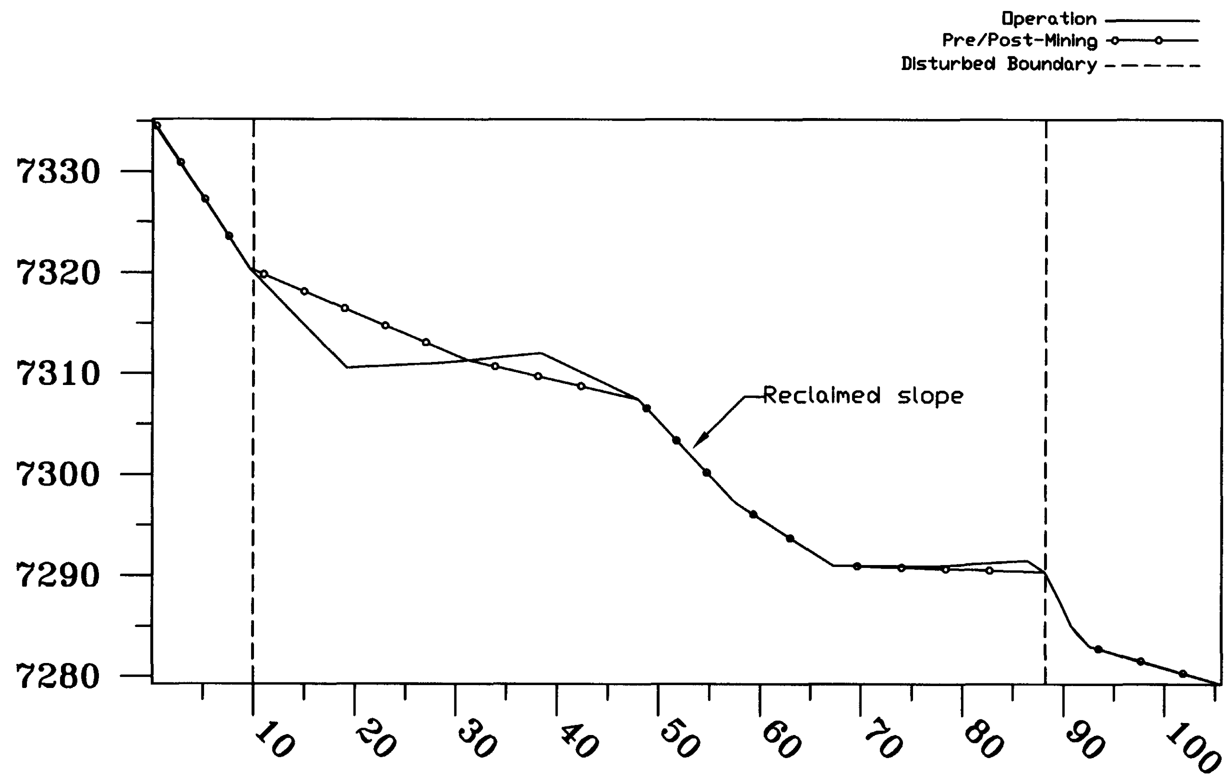
TS-6 Section 11+00



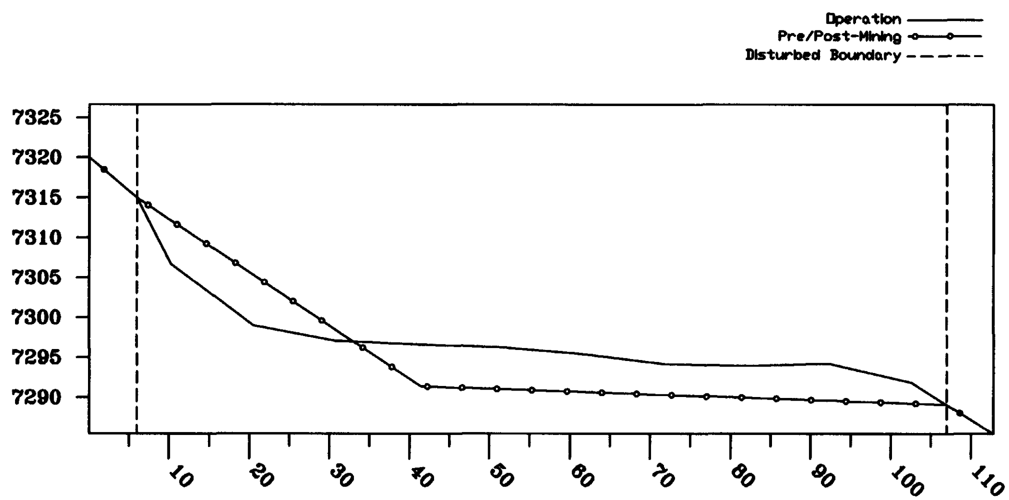
TS-6 Section 12+00



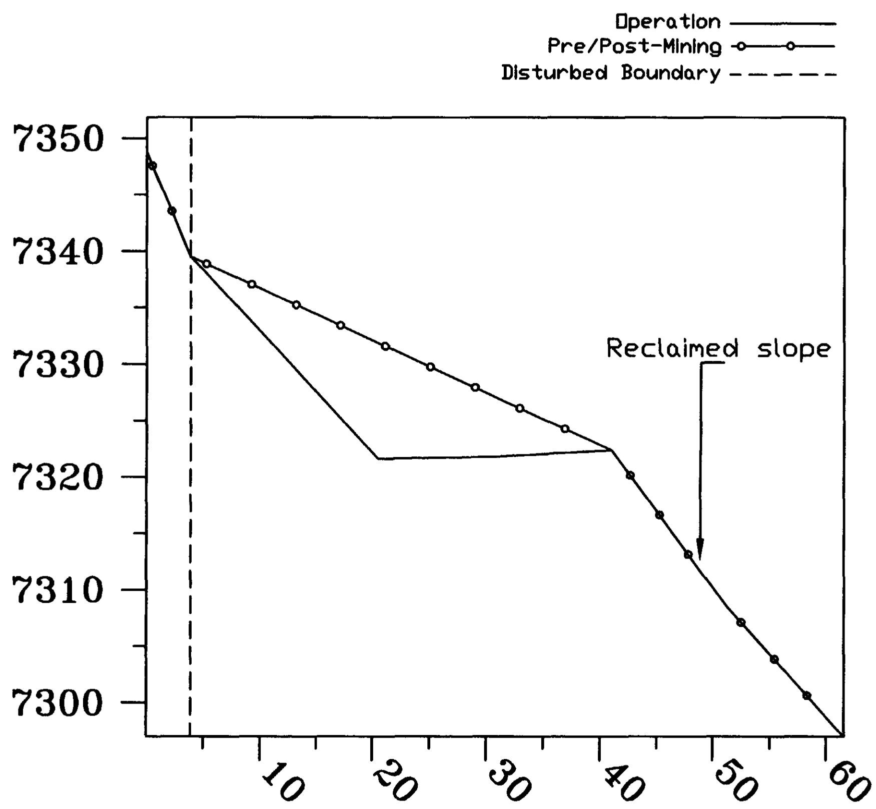
TS-6 Section 13+00



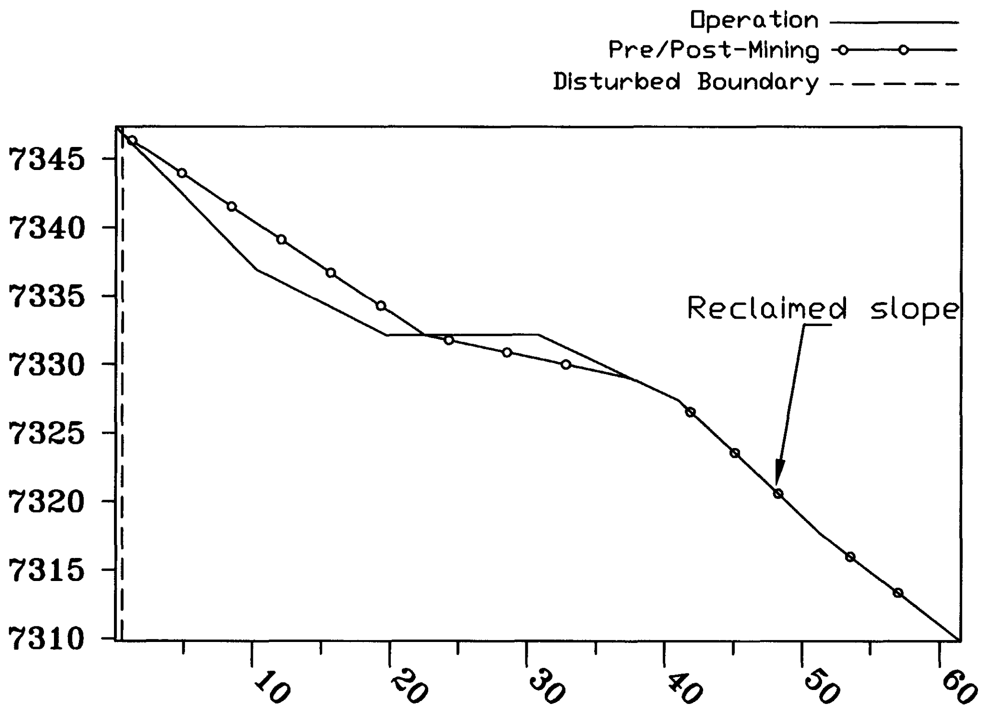
TS-6 Section 14+00



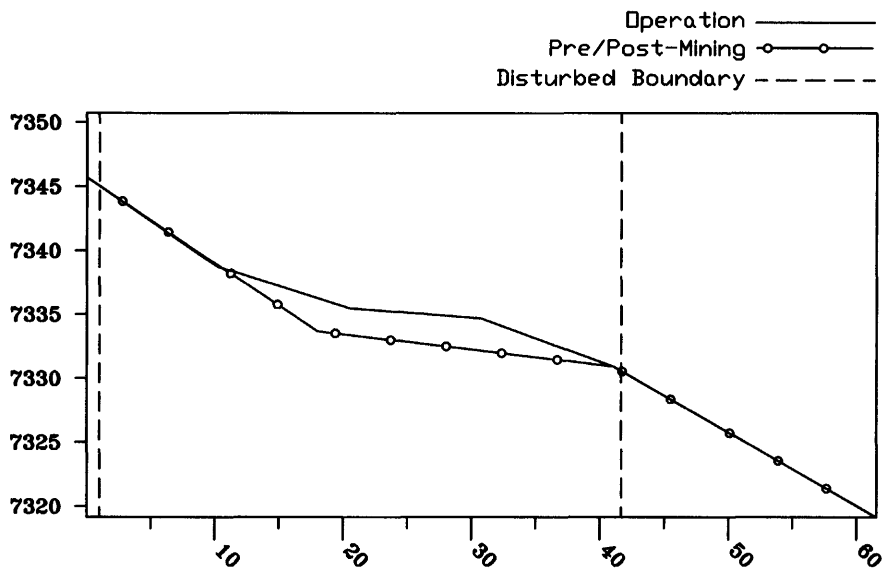
TS-6 Section 15+00



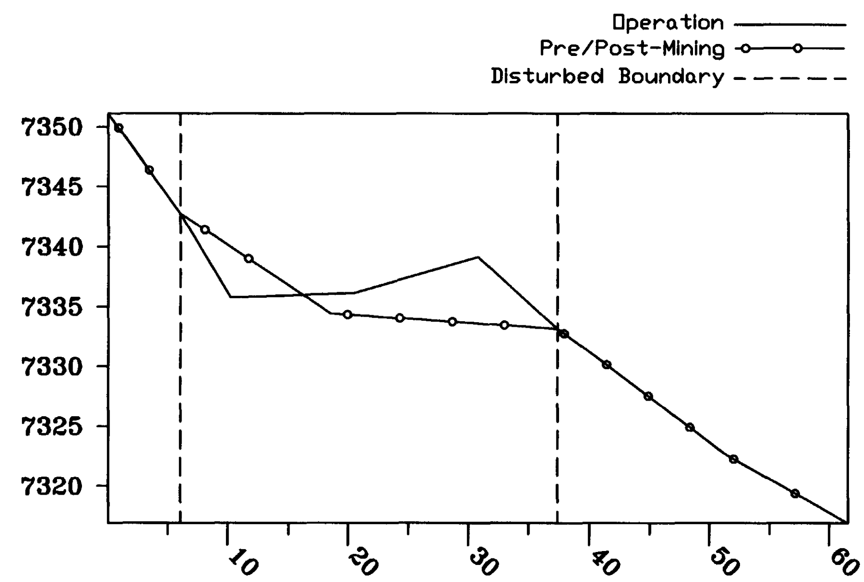
TS-6 Section 16+00



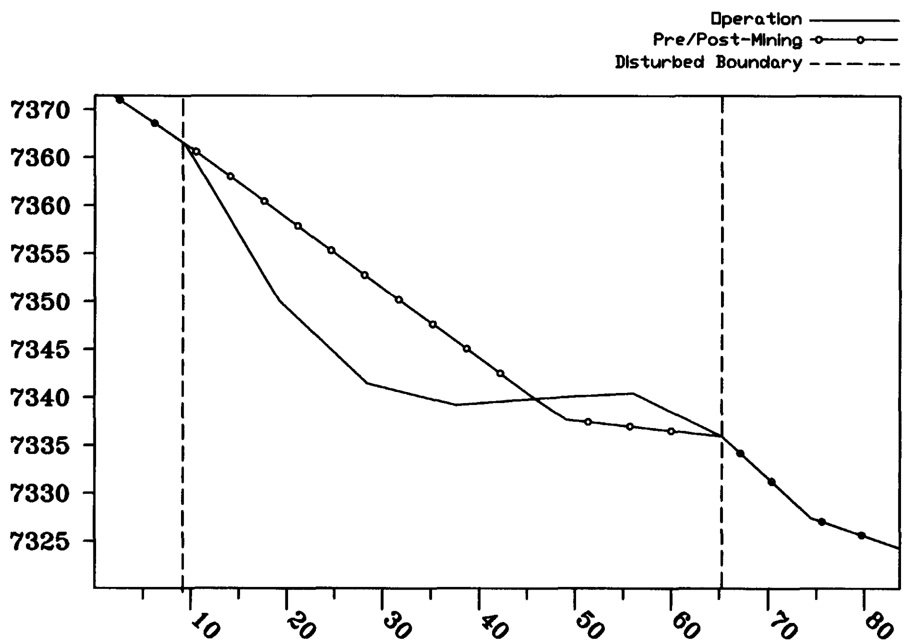
TS-6 Section 17+00



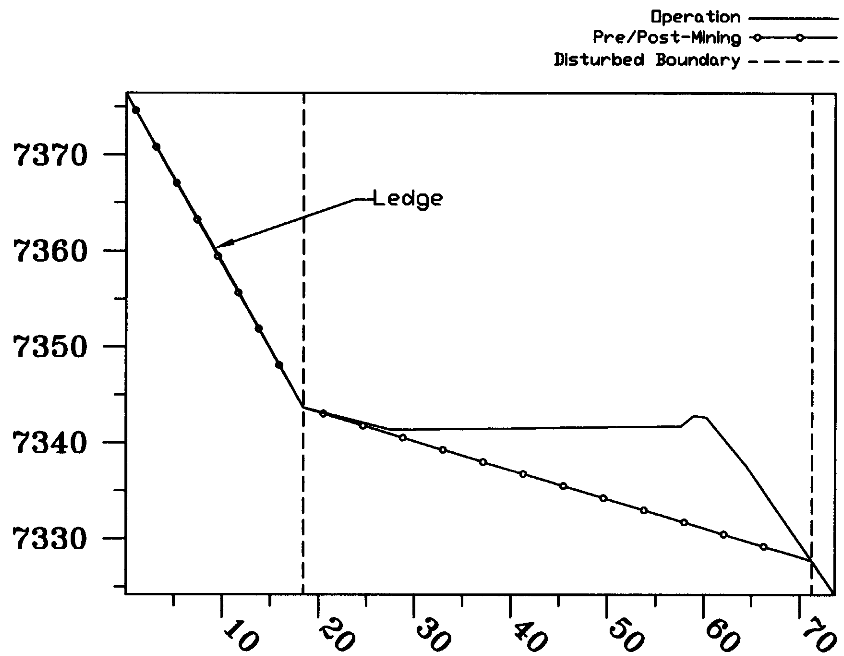
TS-6 Section 18+00

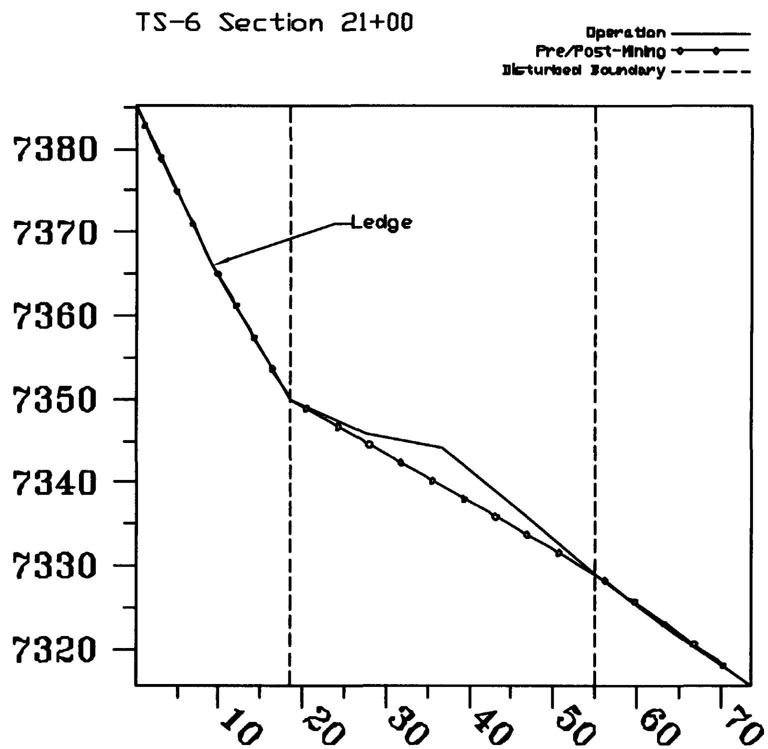


TS-6 Section 19+00

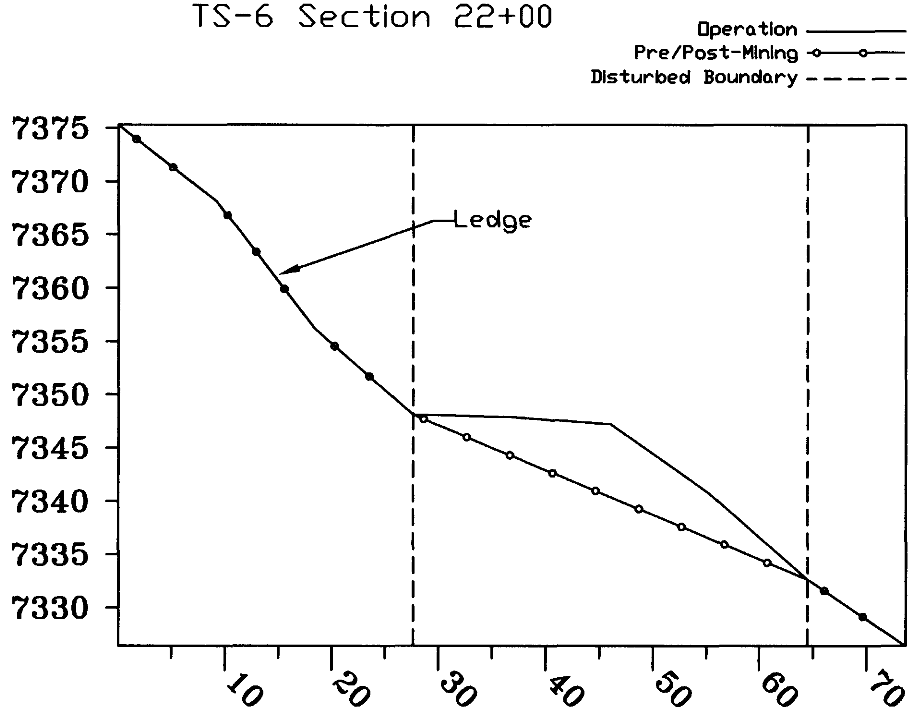


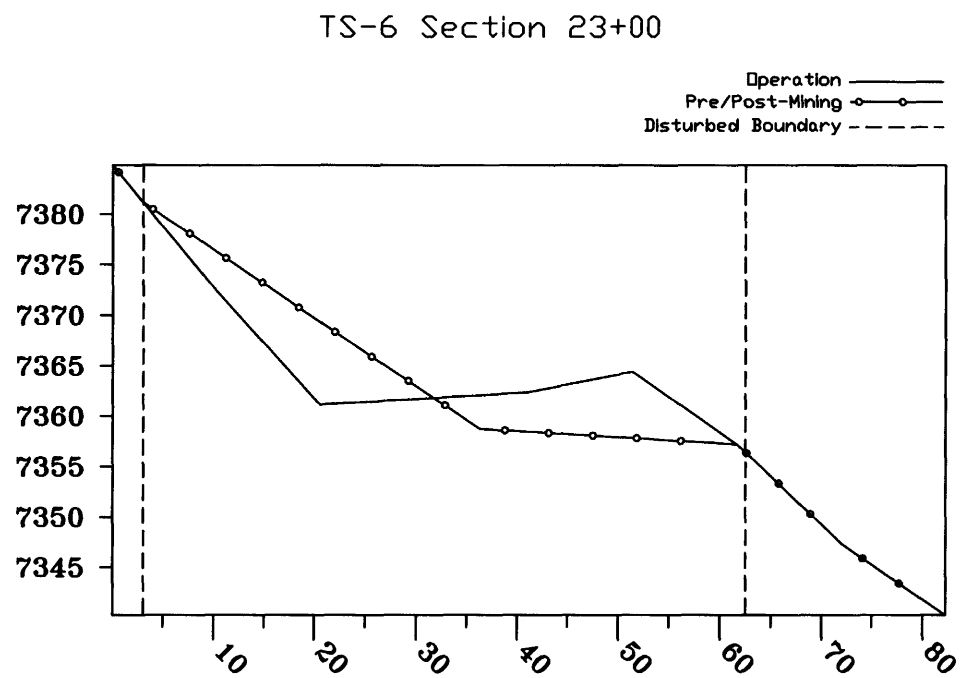
TS-6 Section 20+00



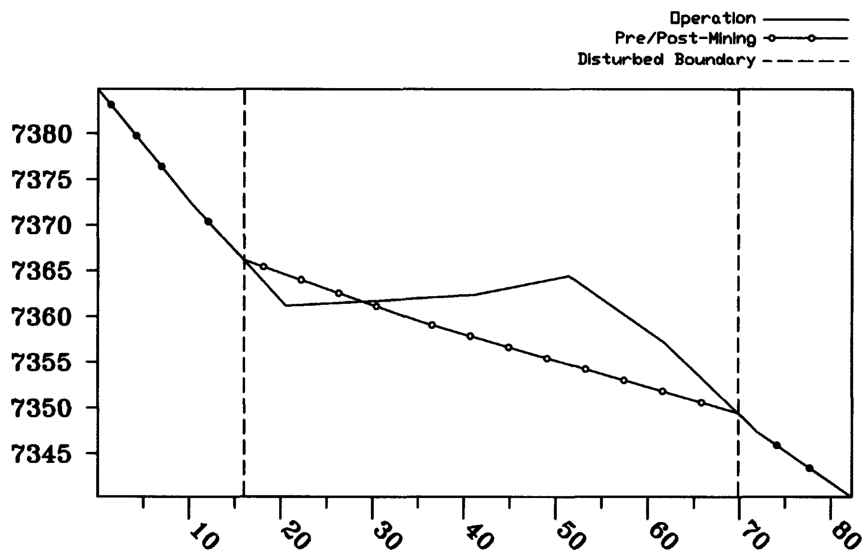

TS-6 Section 21+00
Operation
Pre/Post-Mining
Disturbed Boundary
Ledge
7380
7370
7360
7350
7340
7330
7320
10
20
30
40
50
60
70

TS-6 Section 22+00

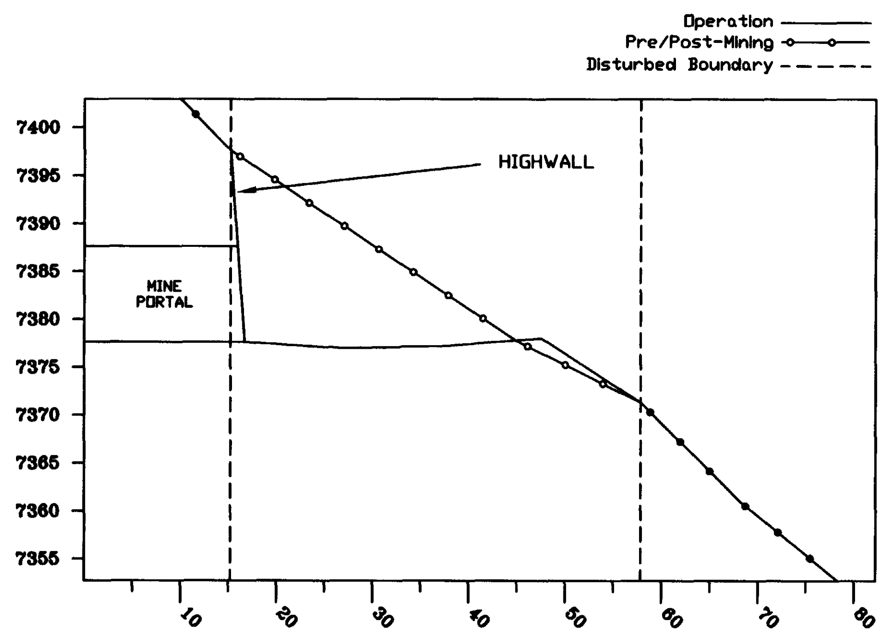



TS-6 Section 23+00
Operation
Pre/Post-Mining
Disturbed Boundary
7380
7375
7370
7365
7360
7355
7350
7345
10
20
30
40
50
60
70
80

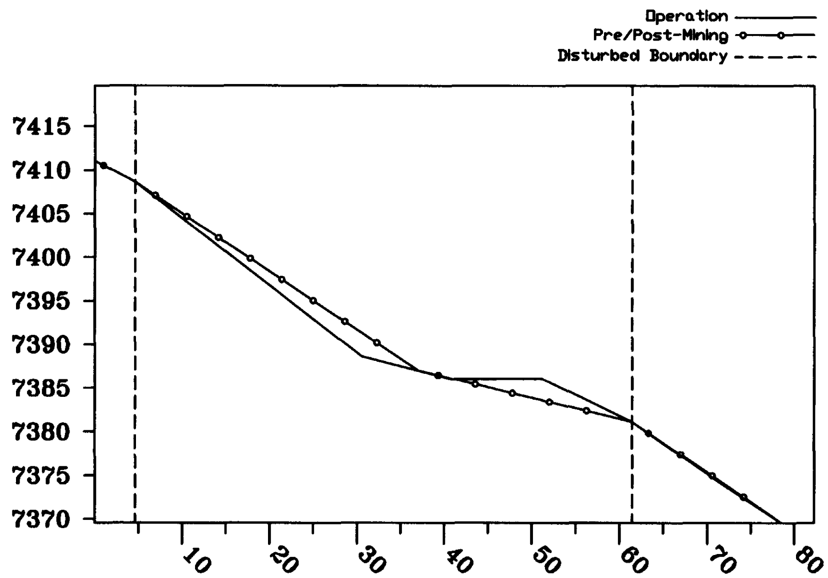
TS-6 Section 23+50

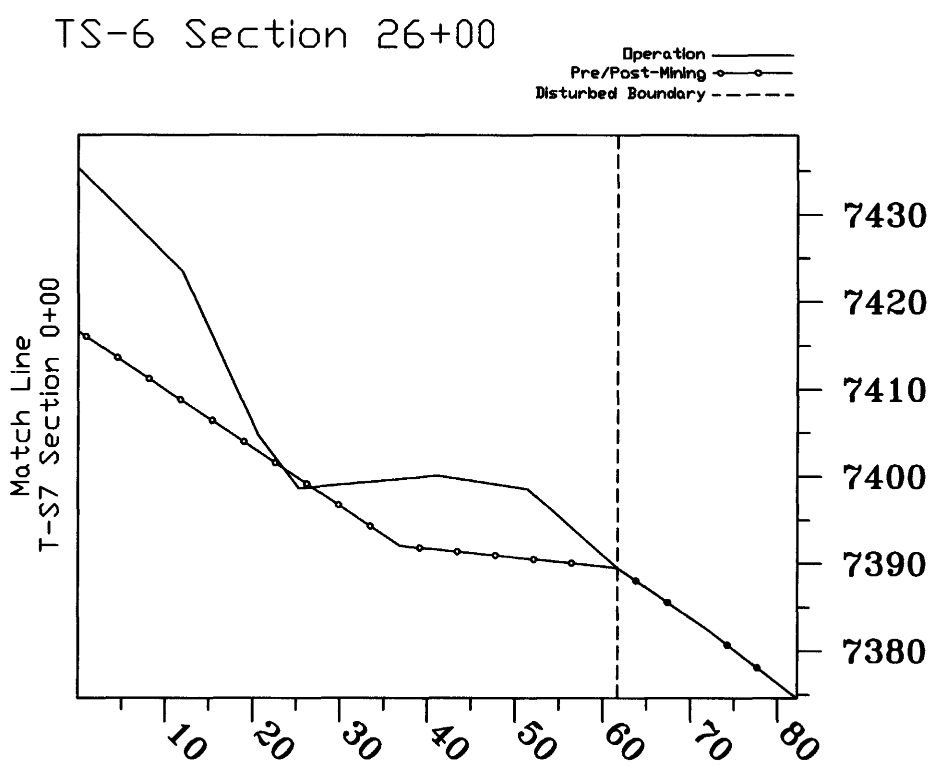


TS-6 Section 24+00

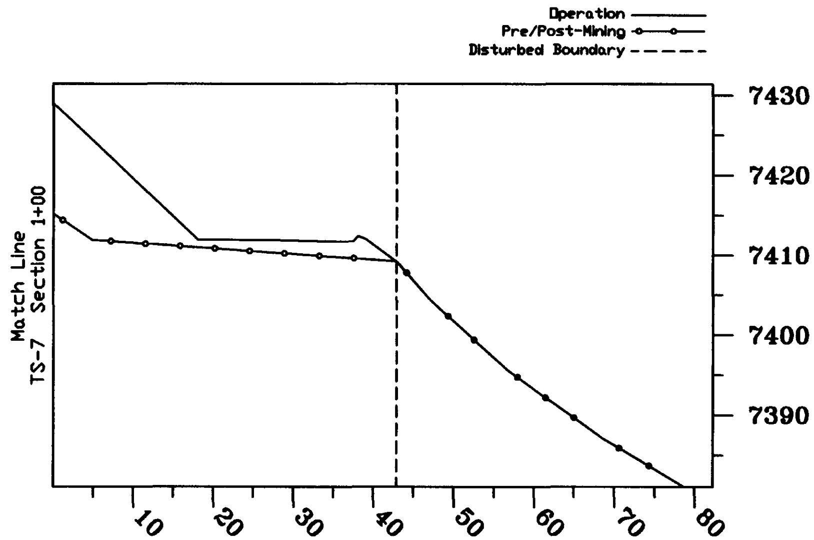


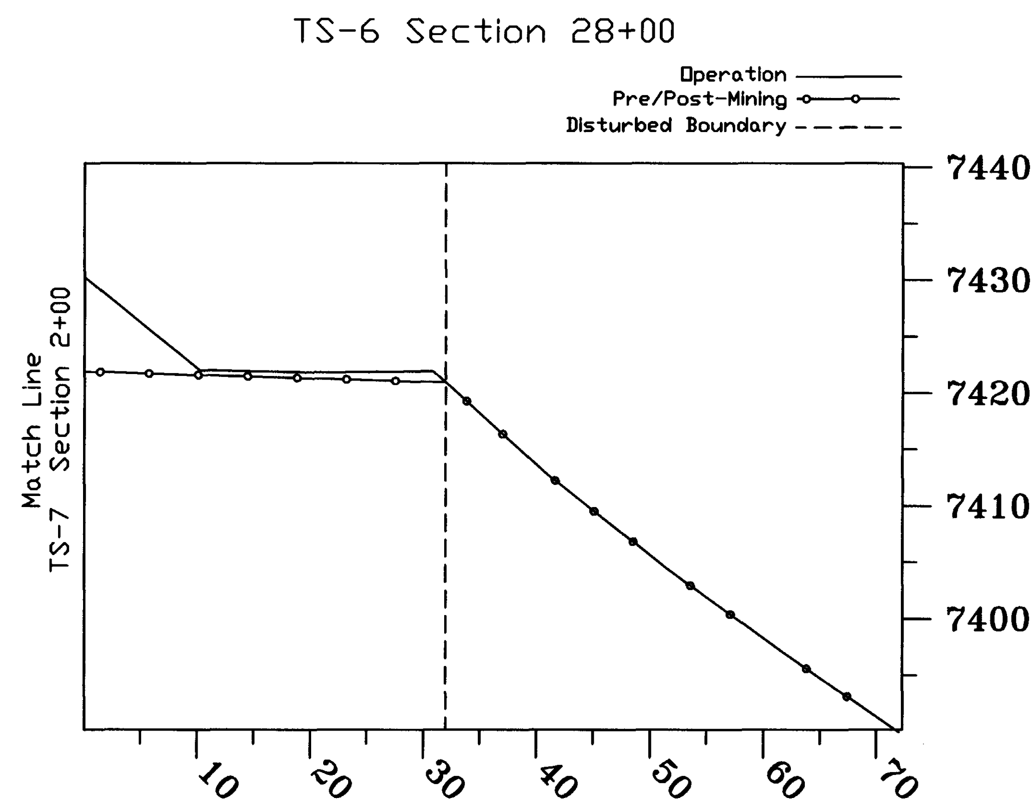
TS-6 Section 25+00




TS-6 Section 26+00
Operation
Pre/Post-Mining
Disturbed Boundary
Match Line
T-S7 Section 0+00
7430
7420
7410
7400
7390
7380
10
20
30
40
50
60
70
80

TS-6 Section 27+00




TS-6 Section 28+00
Operation
Pre/Post-Mining
Disturbed Boundary
Match Line
TS-7 Section 2+00
7440
7430
7420
7410
7400
10
20
30
40
50
60
70

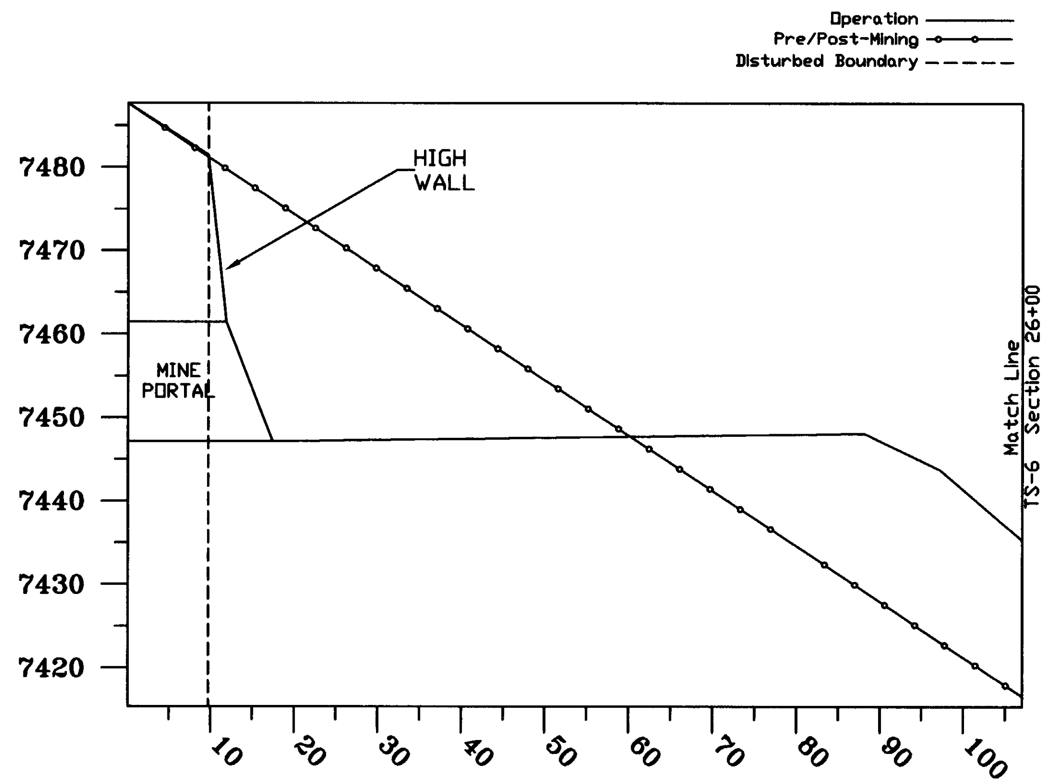
TS-7 Blind Seam Portal Pad

TS-7 will be reclaimed as shown on the following cross-sections in order to match the contours shown on plate 5-6C. 11,582 cu. yd. of material from either TS-5 or TS-6 will be used here for the reclamation. Three highwalls are located in this section and all of them will be completely covered with fill material. The highwall shown on section 3+00 is the belt entry and passes under the road before it enters the coal seam. Table 5I-6 show a summary of the cut and fill volumes.

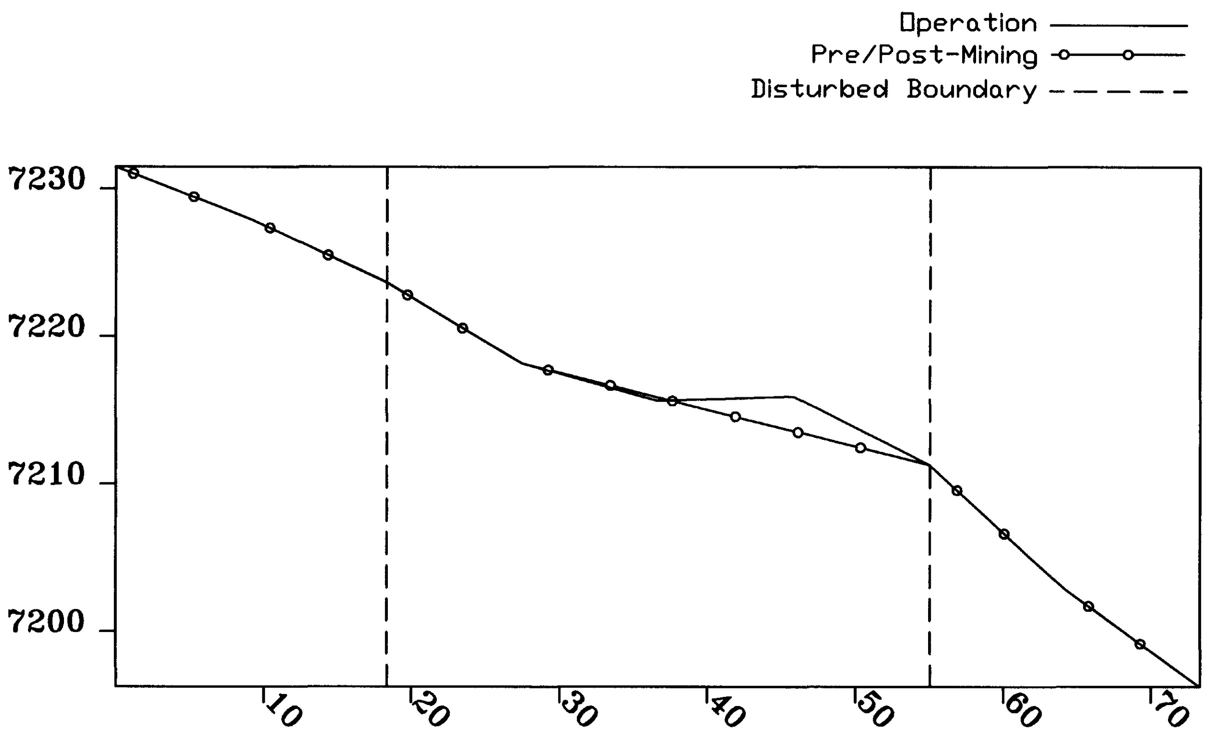
Table 5I-6 - Area TS-7 Cut & Fill Summary

	Fill (-) Volumes (cu. yd.)	Cut (+) Volumes (cu. yd.)			**Volume Cumulative (cu. yd.)**
Section	**Total Fill Volume**	Substitute Topsoil	Regular Soil	**Total Cut Volume**	
0+00	**2,881**	1,078	1,274	**2,352**	**-529**
1+00	**2,633**	867	718	**1,585**	**-1,577**
2+00	**1,630**	578	274	**852**	**-2,355**
3+00	**1,356**	248		**248**	**-3,463**
3+50	**1,355**	137		**137**	**-4,681**
4+00	**1,996**	45		**45**	**-6,632**
5+00	**1,703**	685		**685**	**-7,650**
6+00	**696**	67		**67**	**-8,279**
7+00	**989**	420		**420**	**-8,848**
7+50	**443**	0		**0**	**-9,291**
8+00	**2,355**	45	19	**64**	**-11,582**
Totals	**18,037**	4,170	22,207	**6,455**	

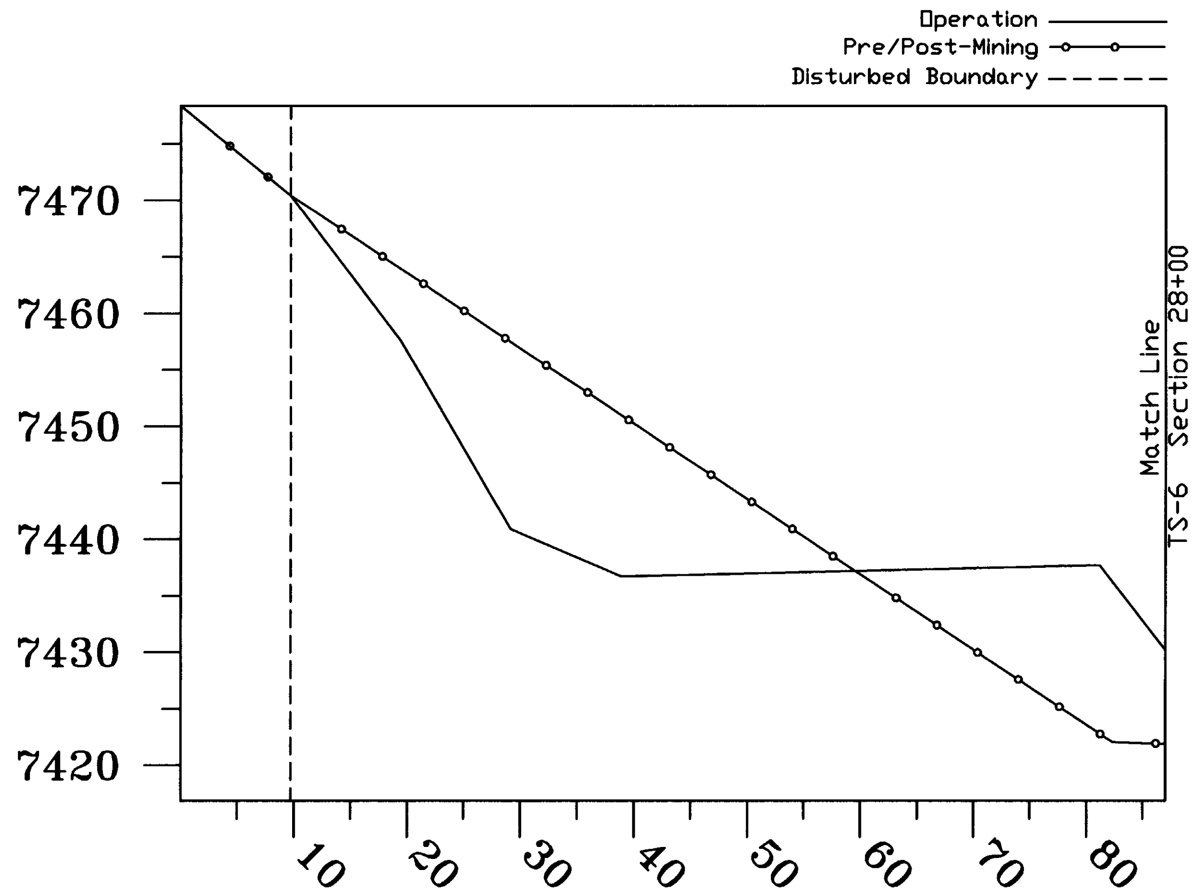
TS-7 Section 0+00



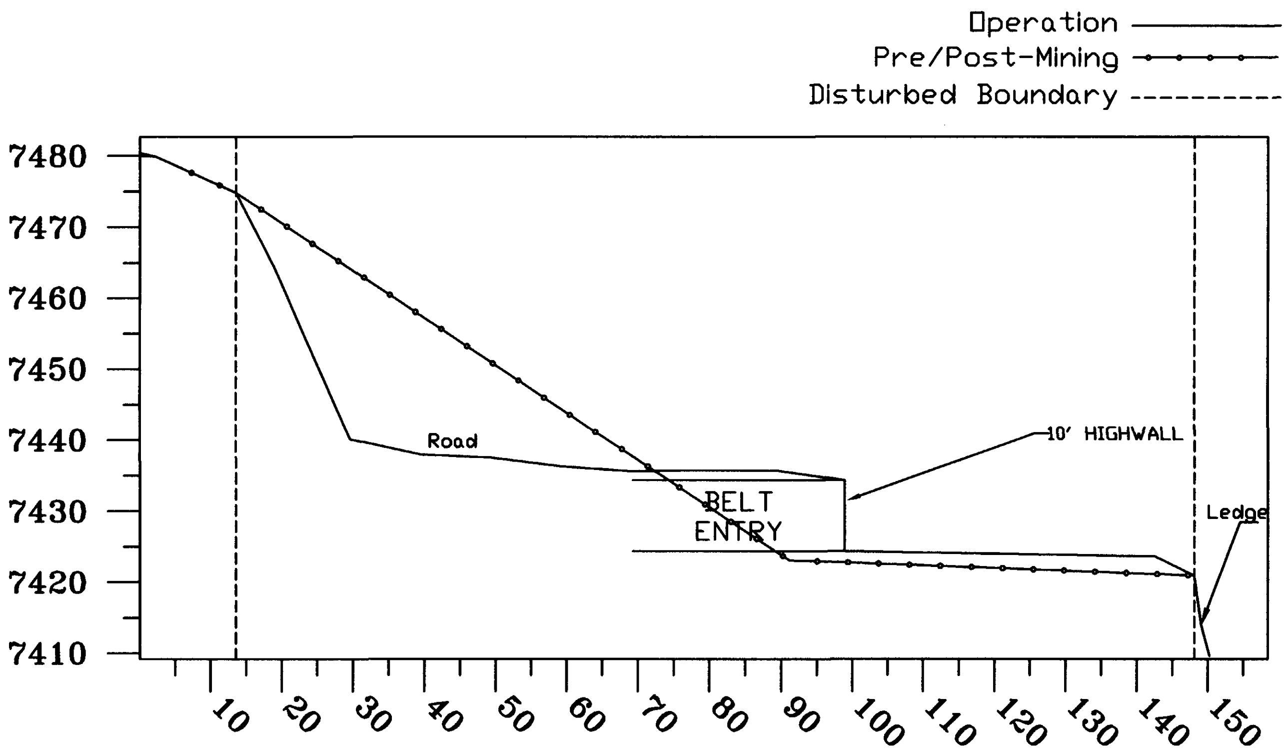
TS-6 Section 1+00



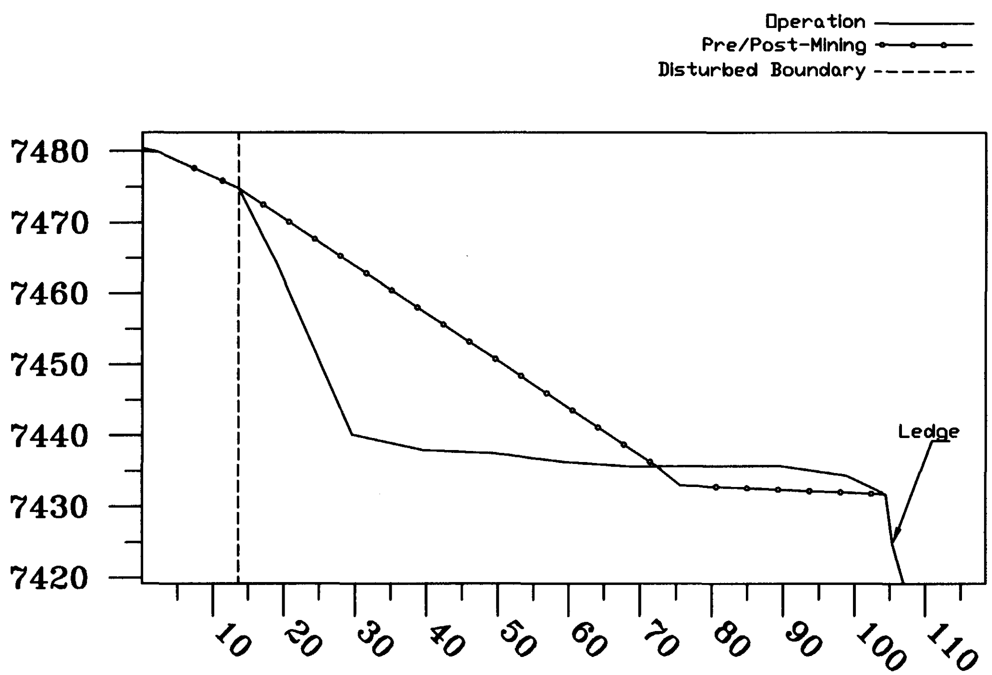
TS-7 Section 2+00



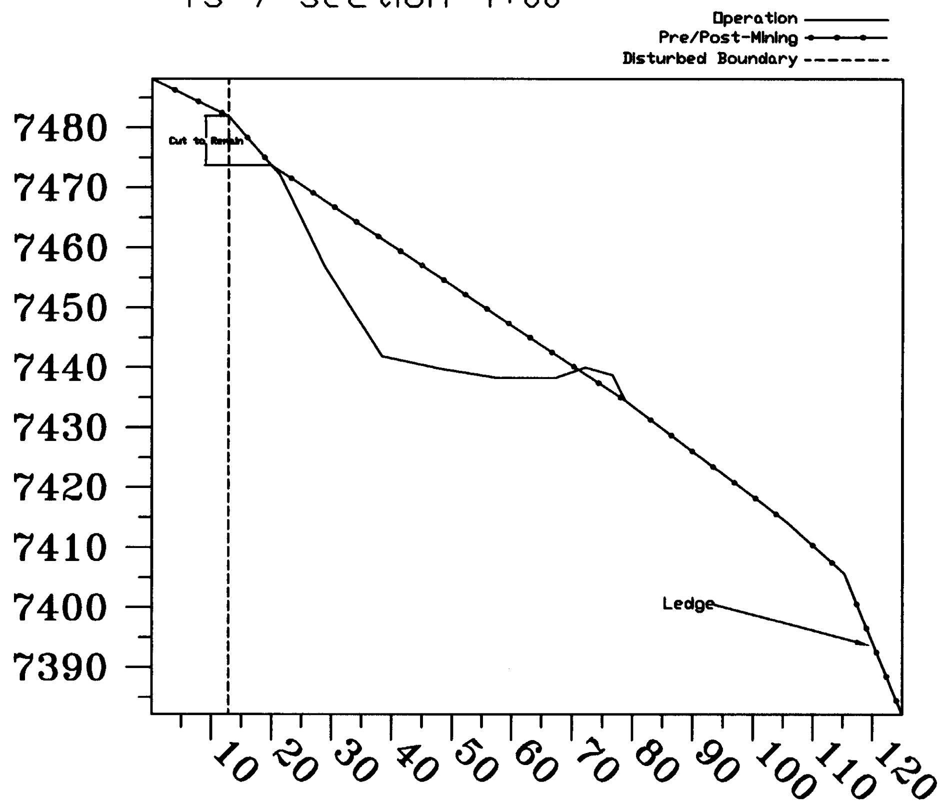
TS-7 Section 3+00



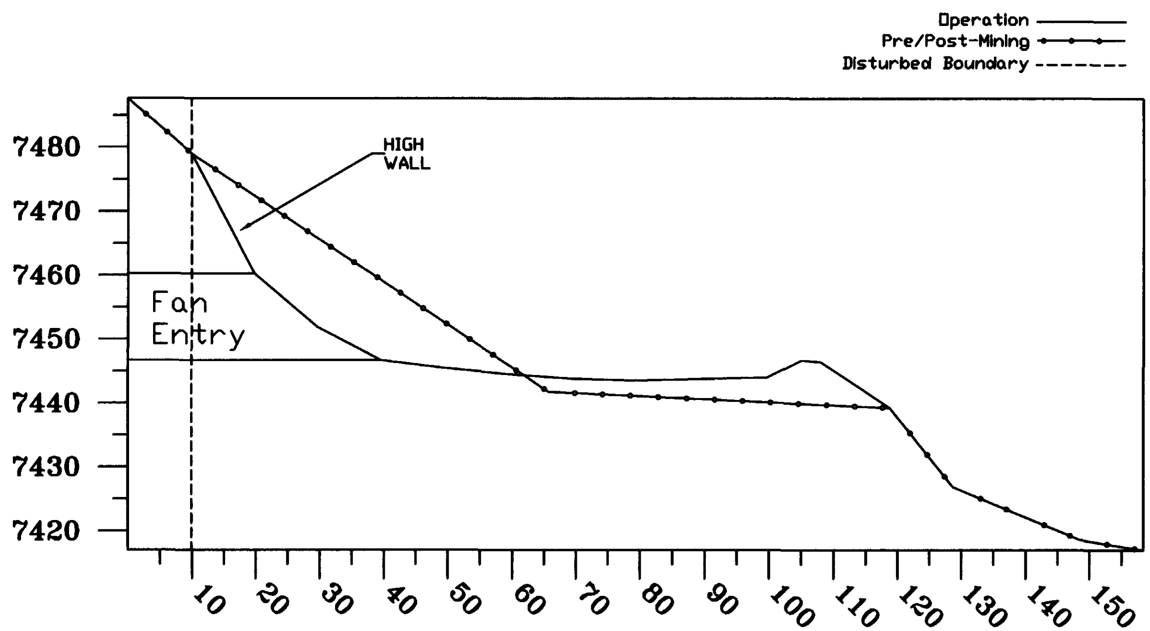
TS-7 Section 3+50



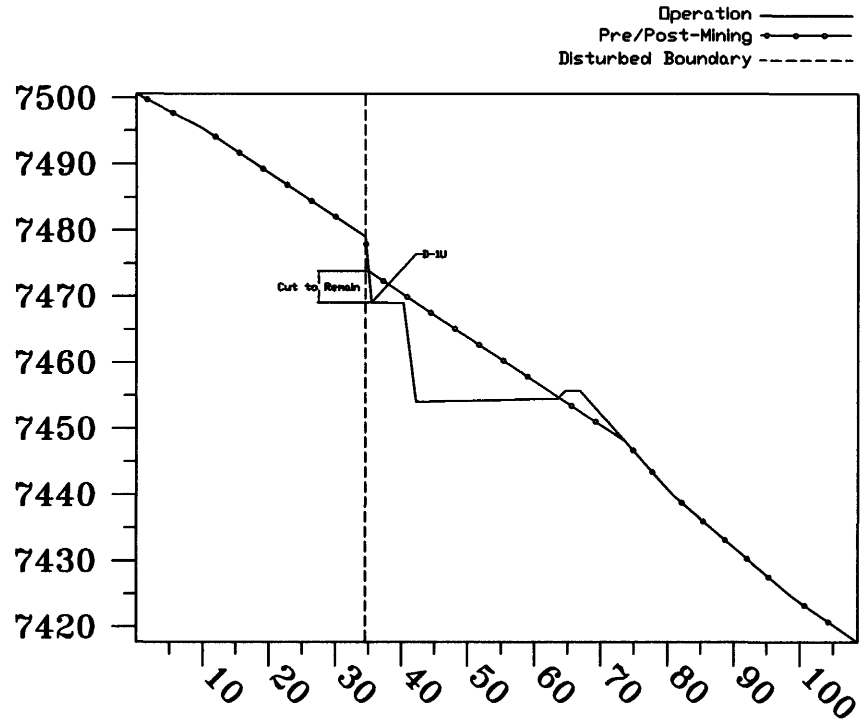
TS-7 Section 4+00



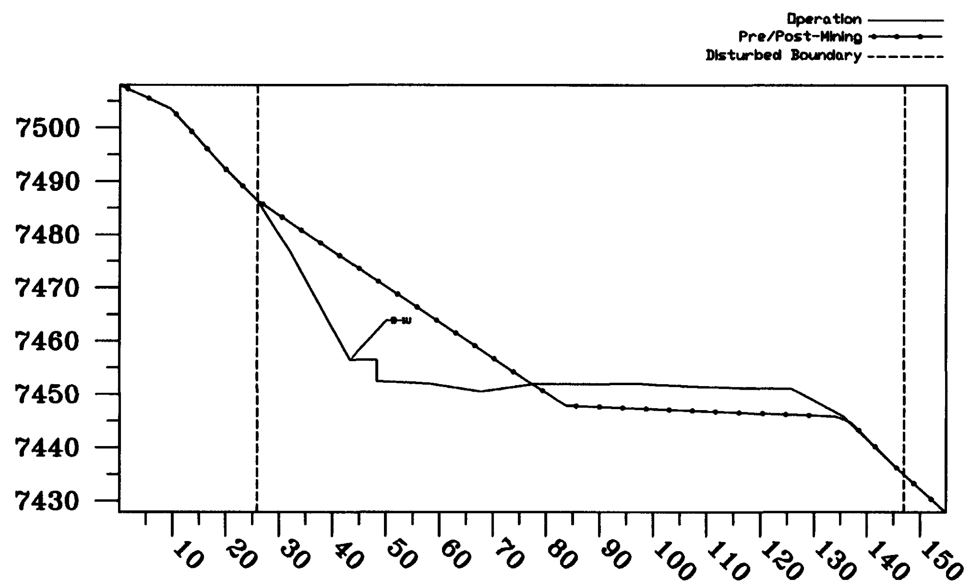
TS-7 Section 5+00



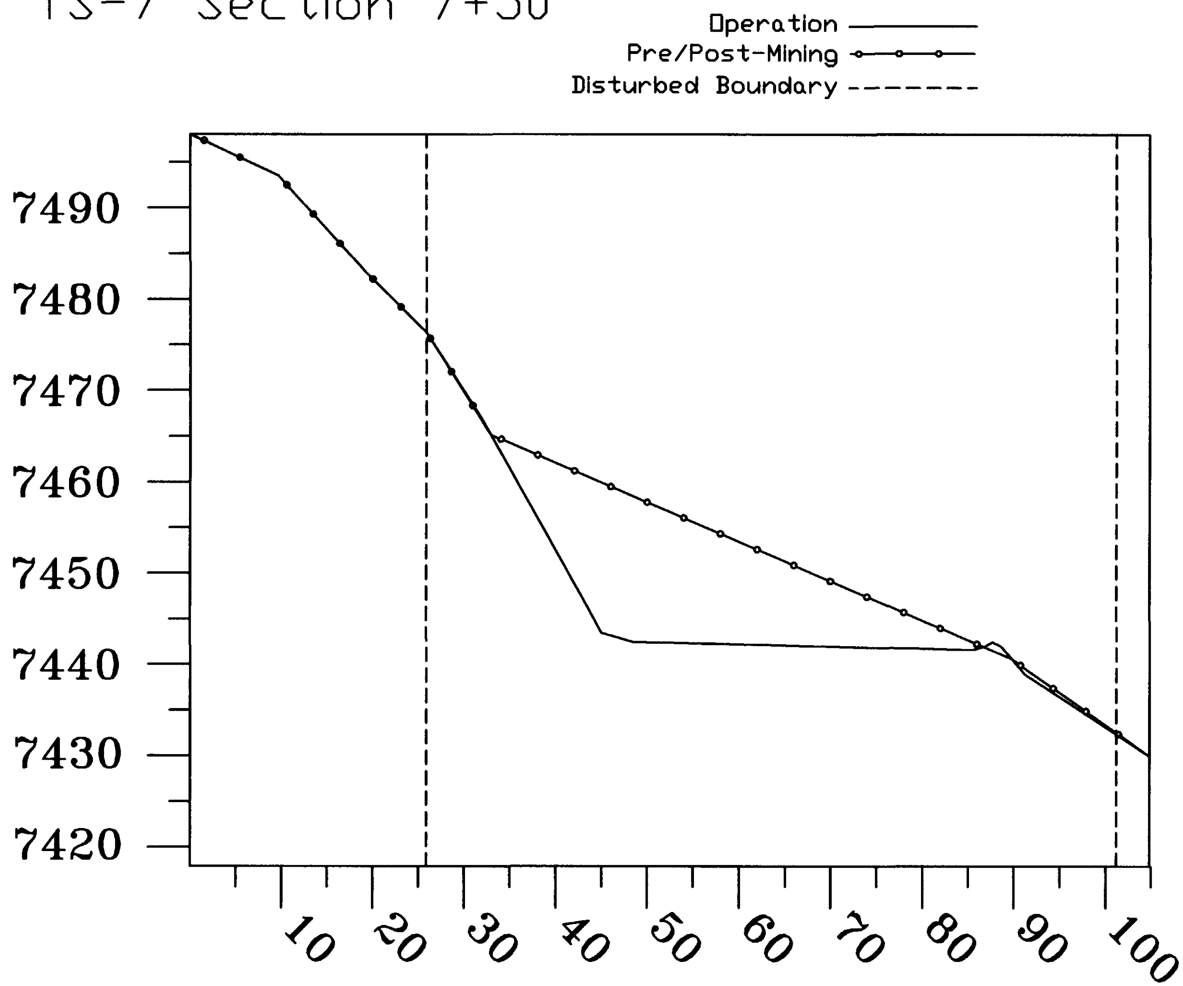
TS-7 Section 6+00



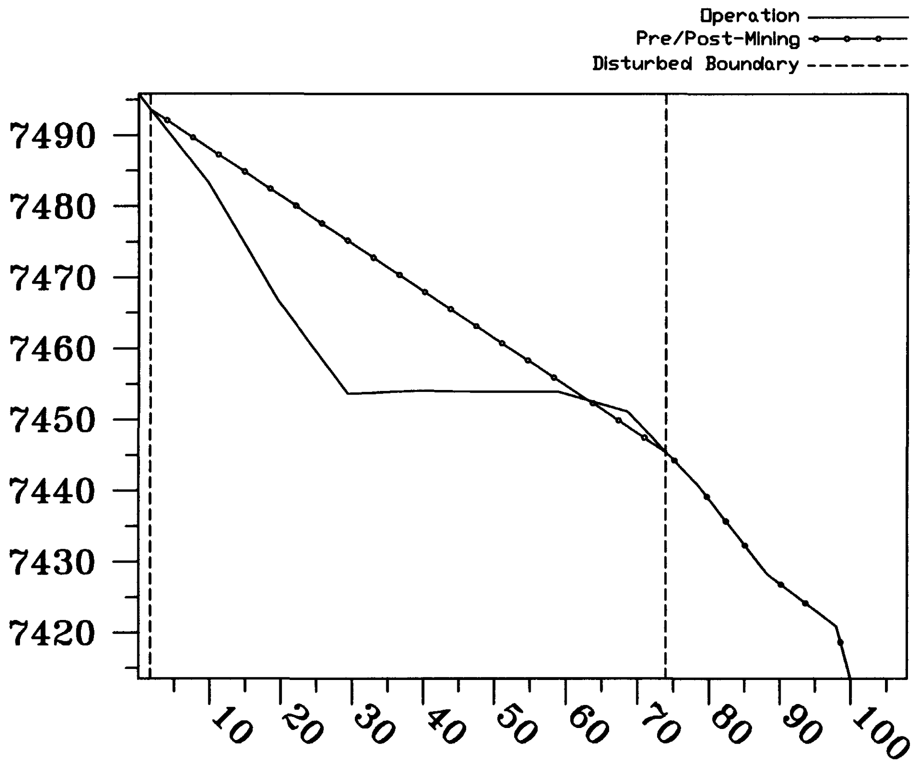
TS-7 Section 7+00



TS-7 Section 7+50



TS-7 Section 8+00



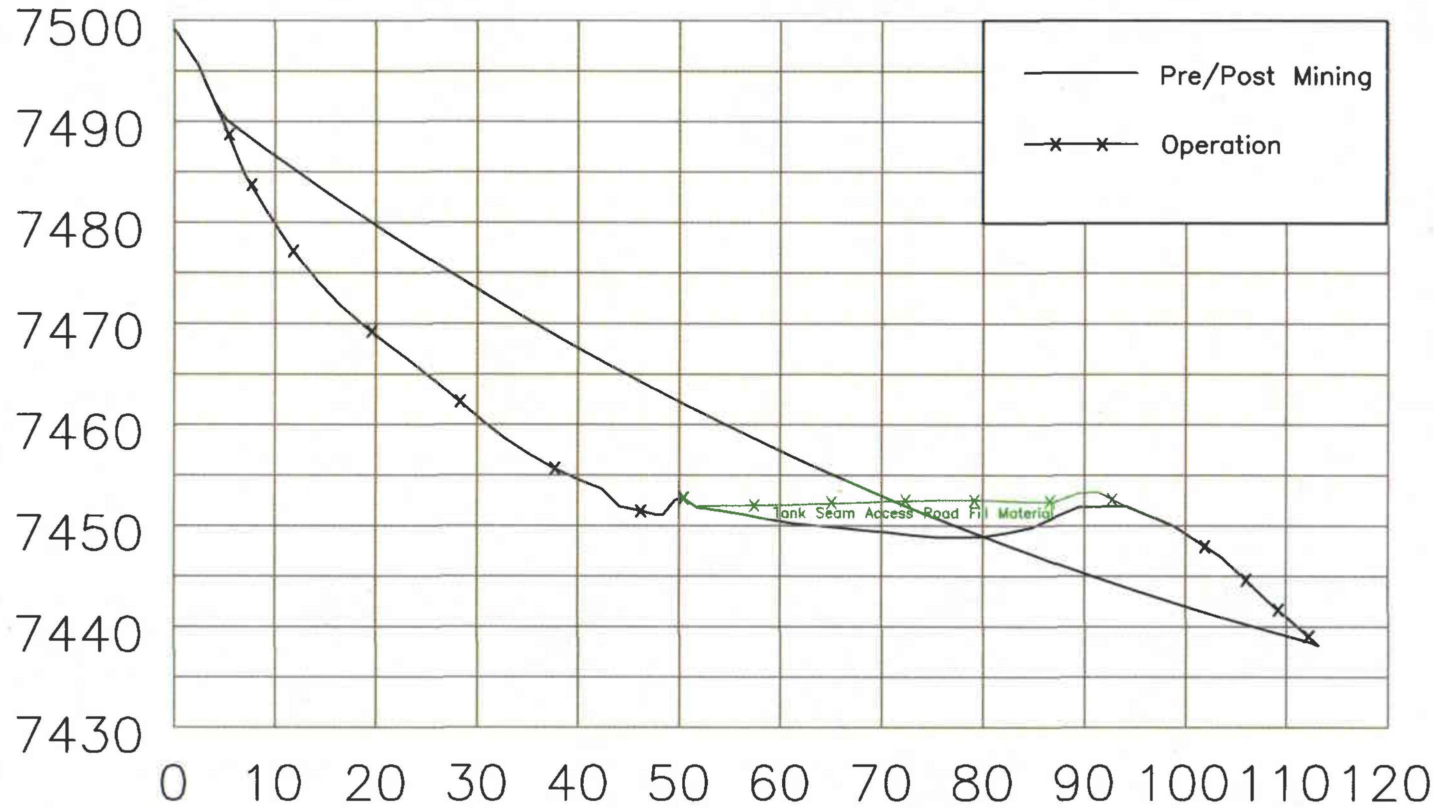
TS-8 Upper Storage Pad

TS-8 will be reclaimed as shown on the following cross-sections. The soil labeled as Tank Seam Access Road fill material was not included in the calculations since it will be used while reclaiming the Tank Seam Access Road (Appendix 5-G). 1,000 yds^3 of this material will remain in place as described on page 5G-10. A volume of 952 cu. yd. of fill material will come from TS-5 or TS-6. A summary of the cut and fill volumes is shown in Table 5I-7.

Table 5I-7 - Area TS-8 Cut & Fill Summary

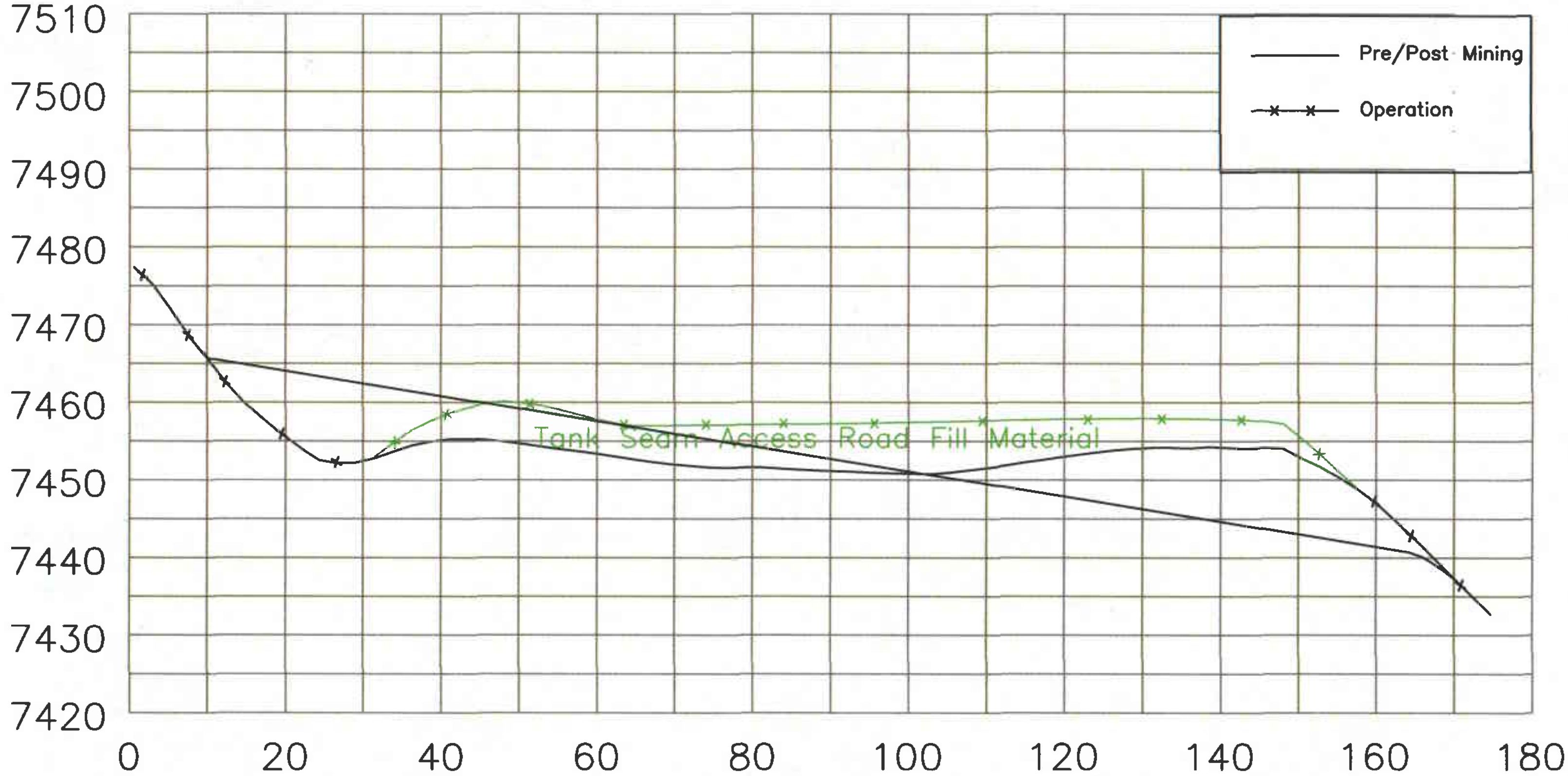
	Fill (-) Volumes (cu. yd.)	Cut (+) Volumes (cu. yd.)			**Volume Cumulative (cu. yd.)**
Section	**Total Fill Volume**	Substitute Topsoil	Regular Soil	**Total Cut Volume**	
0+00	**2,100**	552	0	**552**	**-574**
1+00	**1,300**	1,463	7	**1,470**	**-404**
2+00	**2,218**	1,537	107	**1,644**	**-1,952**
Totals	**5,618**	3,552	114	**3,666**	

TS-8 Section 0+00
Upper Storage Area

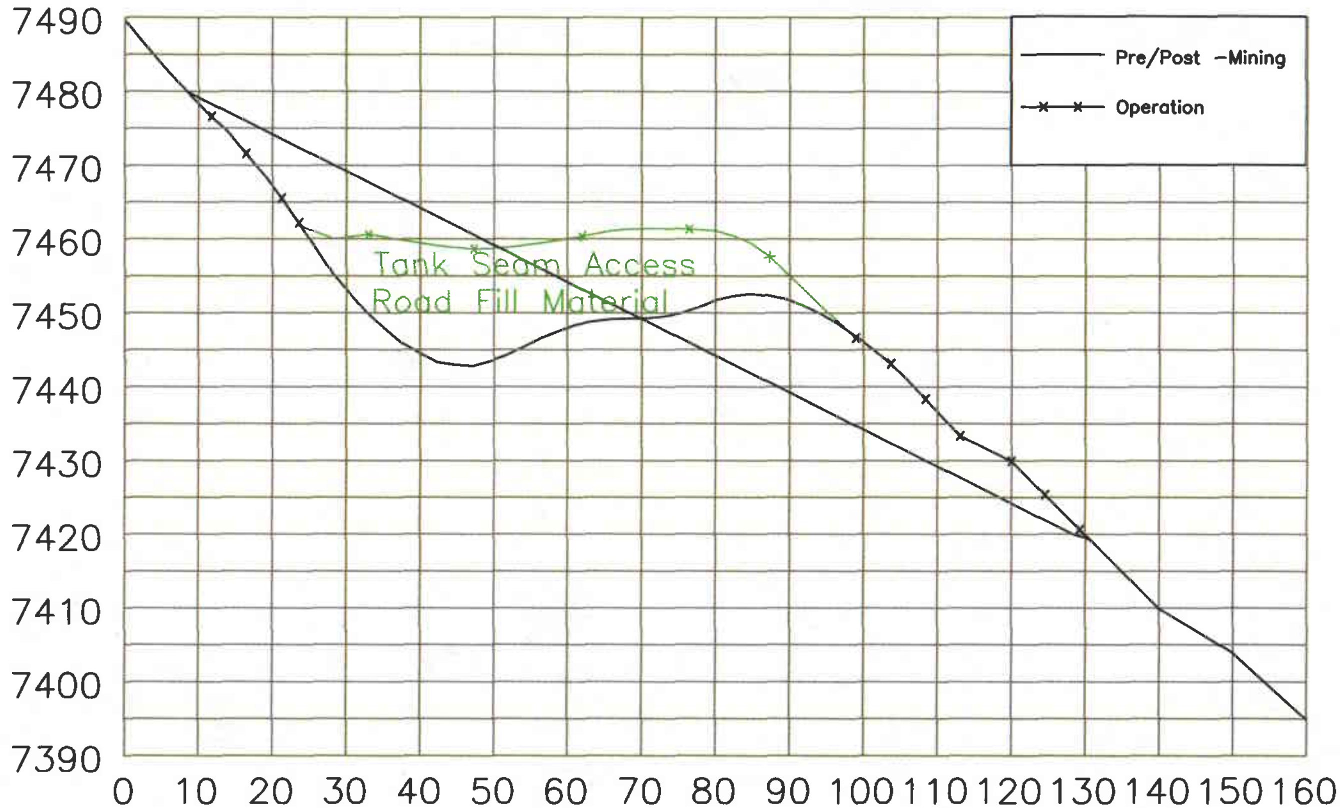


TS–8 Section 1+00

Upper Storage Pad



TS-8 Section 2+00
Upper Storage Pad



TS-9 Sediment Pond C and Bathhouse Pad

The material generated for the bathhouse parking area will be used as fill material for Sediment Pond C and the ditch leading from the Bathhouse Pad to Sediment Pond C. The 1,200 cu. yd. topsoil stockpile created during the construction of the bathhouse pad will be used in conjunction with the substitute topsoil generated from the bathhouse pad.

Table 5I-8 - Area TS-9 Cut & Fill summary

	Fill (-) Volumes (cu. yd.)			Cut (+) Volumes (cu. yd.)			**Volume Cumulative (cu. yd.)**
Section	Topsoil	Substitute Topsoil	Regular Soil	Topsoil	Substitute Topsoil	Regular Soil	
D-D	1,200*	1,762	2,899*	1,200	2,561	2,128	**28**

* It was assumed that sediment Pond C would contain 98 cu. yd. of sediment at the start of reclamation. 1,200 cu. yd. of material will come from the Wild Horse Ridge topsoil stockpile, which was originally recovered from the Bathhouse Pad.

SECTION D–D

SEDIMENT POND "C"